MINING ENGINEERING

AUGUST 1949

Mineralogical Methods in Mineral Exploration

•

Stock Piling

•

Western Phosphate Symposium

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Flotation of Quartz

A MONTHLY PUBLICATION OF THE
AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS
MINING BRANCH

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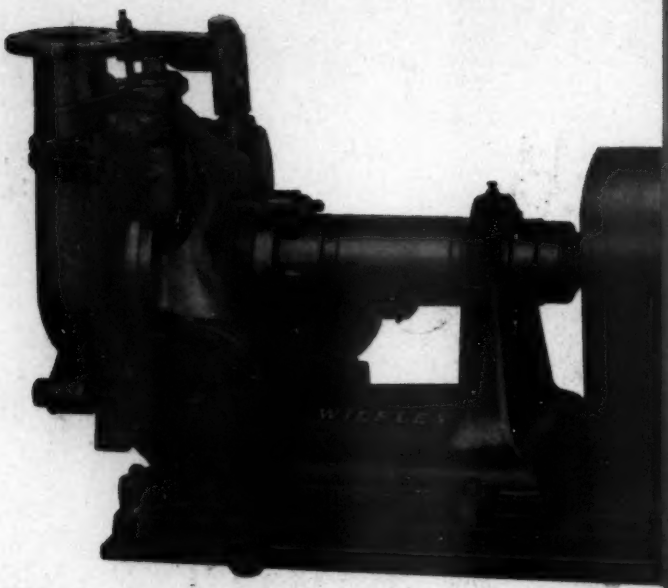

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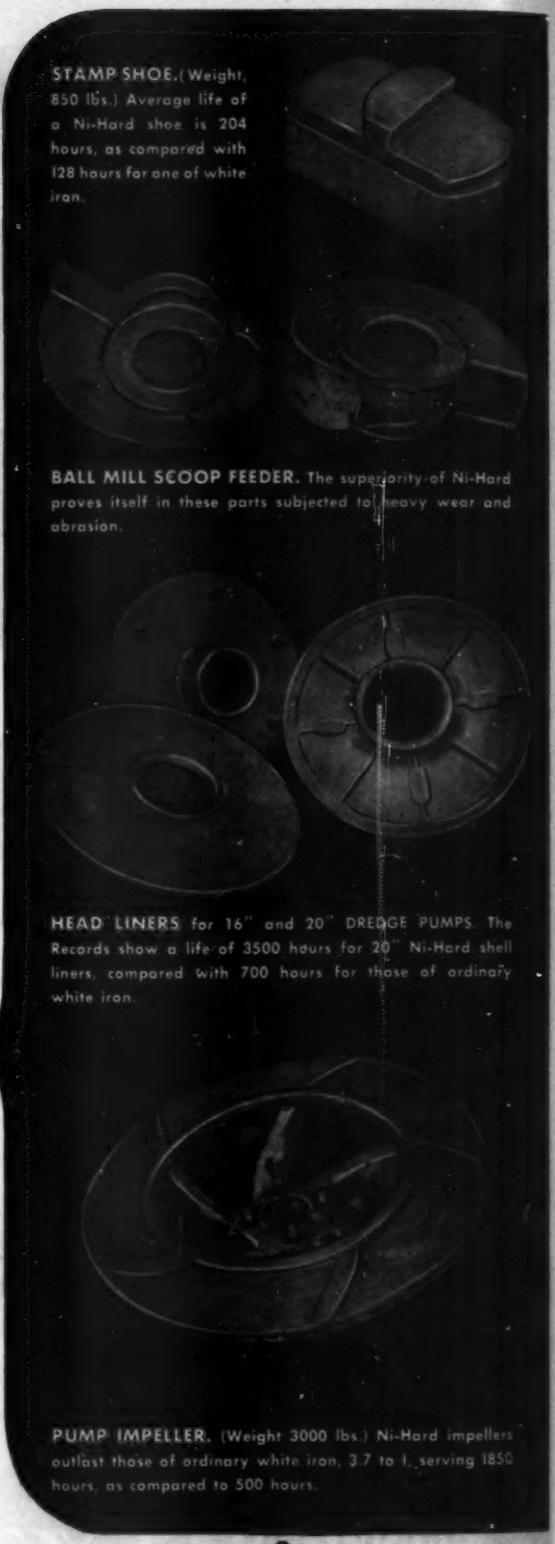

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MINING ENGINEERING

In which is incorporated Mining and Metallurgy, Mining Technology and Coal Technology

AUGUST 1949 VOLUME 1, No. 8

On the cover: Looking down the business end of a rod mill at the Balmat-Edwards division of the St. Joseph Lead Co. Photo by William M. Rittare.

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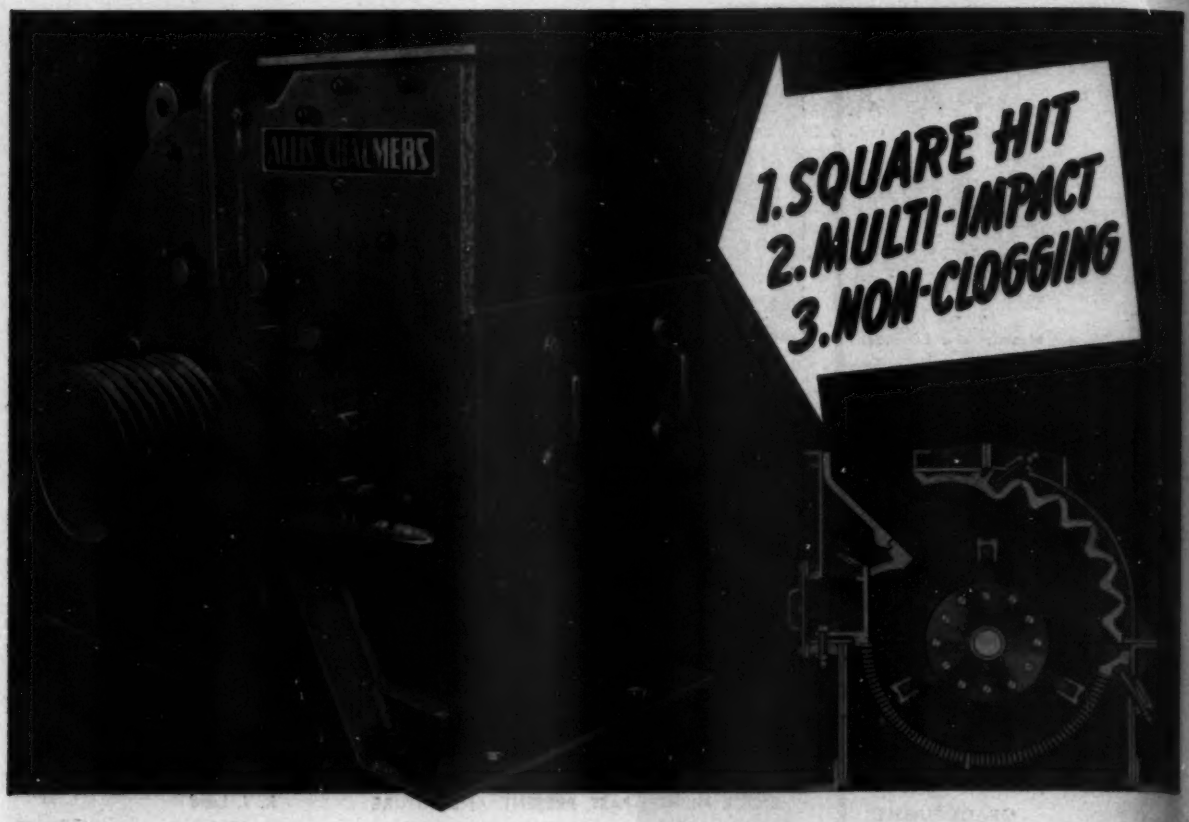

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Iron ore and rock drop from Jalloy Steel lined truck bodies into Jalloy lined chutes. Jalloy chute plates last an entire 6¾-month season compared with 6 weeks service from mild steel plates.

3-year tests prove heat-treated Jalloy plates last 4 to 6 times longer than mild steel under severe abrasion and impact conditions.

Steel men call J&L Jalloy a manganese-molybdenum alloy steel.

But maintenance men for the Inter-State Iron Company on the famed Mesabi Range in Minnesota, call it *"that hard stuff."*

Why? Because they've worked with it for 3 years on some of the toughest applications from the standpoint of abrasion and impact to which steel has ever been subjected.

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They've seen the long life of Jalloy *cut the cost of* steel for certain applications *by more than one-half.*

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For full information send for the booklet mentioned in the coupon below.

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Please send me your booklet: "Jalloy—J&L Alloy Steel."

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Do you recommend Jalloy for_____

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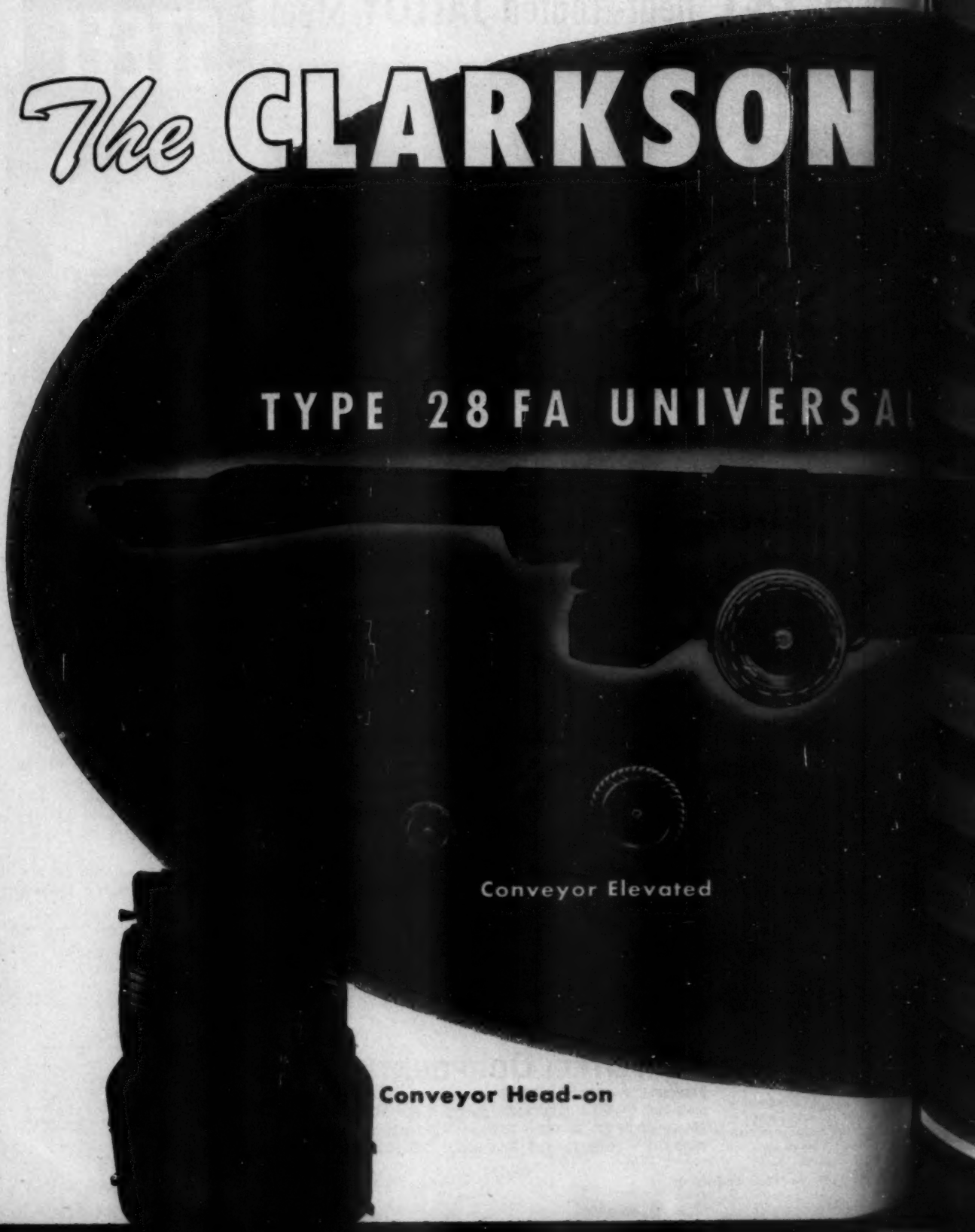

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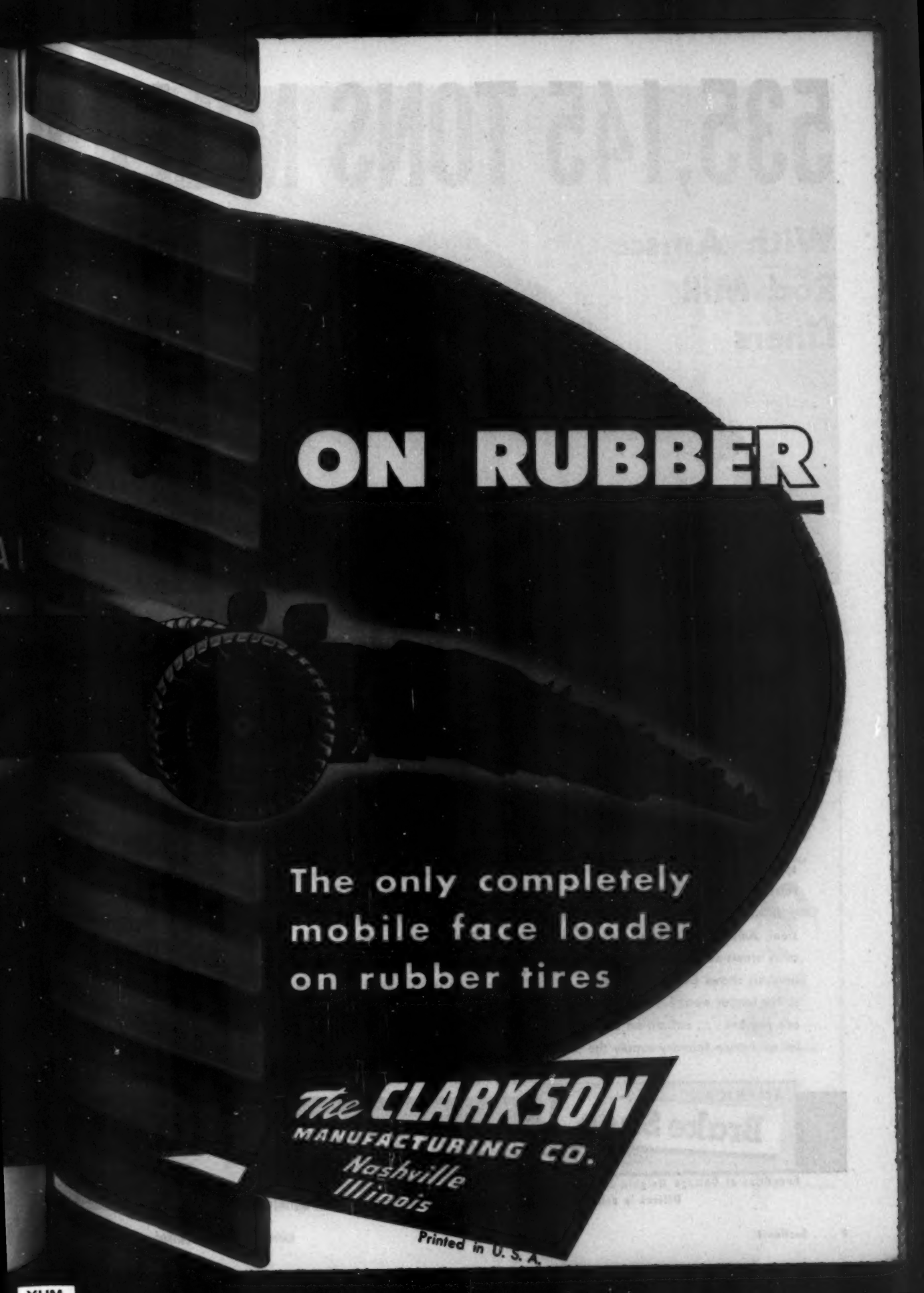
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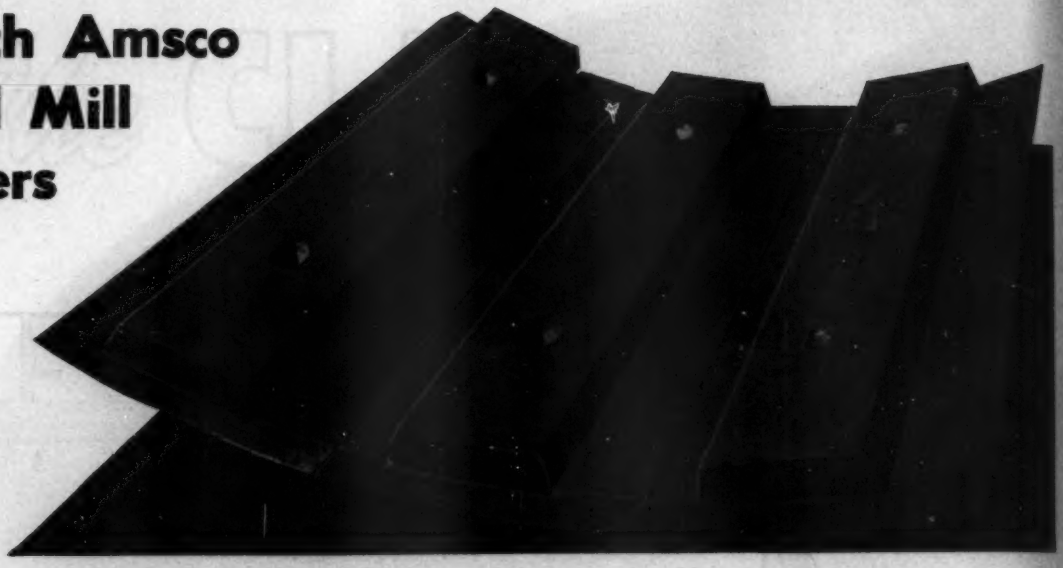
With Amsco Rod Mill Liners



In supplying parts for mining and milling equipment, Amsco gives you an important service in addition to that of providing quality castings. This service is one of metallurgy wherein all the years of specialized Brake Shoe research experience are brought to bear to determine what particular alloy steel will stand up best in your particular application.

The photograph shows an installation where Brake Shoe metallurgical analysis plus the user's design resulted in tremendous savings in operating cost. In this rod mill for handling gold ore, the standard manganese steel liners were replaced only after 535,145 tons of production . . . a record in milling tonnage. In this same installation special manganese steel lifter bars (the third of three sets of lifters) remained effective for 249,100 tons of milling.

In addition to special manganese steel and standard manganese steel, Amsco foundries of course can supply castings made of other alloy steels such as chromium-molybdenum . . . whatever application analysis shows best for the job.

For longer wear life and lower cost, wherever impact and abrasion are present . . . call on an Amsco sales engineer for service . . . and let an Amsco foundry supply the parts.

Parts for rod mills and ball mills are two of many mine and quarry applications for Amsco manganese steels and "chrome-moly" steel. In such mills Amsco liners, grates and feeder lips help toward low cost operation with minimum down time for replacement. For complete information write for the 60 page Bulletin 743-M "Amsco Manganese Steel for Mining and Quarrying Equipment".

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AMERICAN MANGANESE STEEL DIVISION
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Offices in principal cities. In Canada: Joliette Steel Limited, Joliette, Que.

How Utah Screens Solved Tough Soda Ash Sizing Problem . . .

SODA ASH (Na_2CO_3) is produced by burning limestone and salt with ammonia. It's a lumpy, easily crumbled, hard-to-screen material.

A large Midwest processor tried several makes of screens on this job. None were satisfactory. Particle size broke down excessively. Blinding of screen mesh required too much "time out." Maintenance was costly and screen cloth had to be replaced frequently. Then this battery of *Utah* electric screens was installed.

They were immediately successful — and here's why:

- *Finger-tip control* of amplitude of vibration provided close control of operation to suit fine sizing.
- Troublesome blinding, caused by clogging, near-size particles was practically eliminated. No frequent shutdowns to brush screen surfaces.
- Screen body and screen cloth vibrated as a unit . . . uniformly . . . assuring long screen cloth life.
- Maintenance negligible — no flexible reeds, springs or striking parts.

Soda ash is only one of many materials that can be screened successfully with modern *Utah* electro-magnetic vibrating screens. They're flexible enough in operation to handle a wide range of fine, granular materials from 10 to 48 mesh (dry) or 65 mesh (wet).

Utah screens come equipped with a *Vari-Pulse* non-blinding device which periodically increases vibrating amplitude from normal operating setting to maximum, momentarily, to clear screen mesh.

Get more facts about *Utah* screens from the A-C representative in your area. Or write for **Bulletin 07B7044.** Offices or distributors in principal cities in the U.S.A. and throughout the world.

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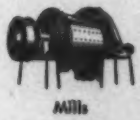

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thru the Eyes of the Metallurgist

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The ultimate economic flow scheme for any one mine may as yet be undetermined. But certain it is that long before high-grade ores run out, the beneficiation and use of low-grade and fine iron ore will be completely practical.

On other beneficiation problems intensive work goes steadily forward in the Cyanamid Mineral Dressing Laboratory . . . problems such as the flotation of certain oxidized Zinc and Copper minerals; the selective flotation of Cassiterite; the economic treatment of Zinc Carbonate, etc., etc.

In Cyanamid Mineral Dressing Laboratory thousands of new and old chemicals are tested; an infinite number of reagent combinations and processes explored. Though only a few significant advances may result from a tremendous amount of technical work, cumula-

SULPHUR

*Interesting Facts Concerning This Basic Raw Material from the Gulf Coast Region

*MOLTEN SULPHUR

The discharge lines from the wells deliver the sulphur into sumps at collecting stations which are located near the area being "steamed."

The sump is dimensioned to suit operating conditions, as well as the number of wells supplying sulphur. Cast iron has been found the most suitable material for lining the sump, and for the steam coils on the bottom and at the sides which keep the sulphur in a liquid state. When the sump is reasonably full, pumps force the liquid sulphur through insulated pipe lines to the vats. The pumps are especially designed for this service, the moving parts being either submerged in liquid sulphur or steam-jacketed.

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This high metallurgical recovery is accomplished in processing approximately 60 tons of ore per 24 hours. A 6-cell No. 15 (24 x 24) Denver "Sub-A" recovers lead and an 8-cell No. 15 (24 x 24) recovers zinc.

The Objective

Because silver is such a valuable constituent of the ore, the flotation objective is to produce a lead concentrate carrying as much silver as possible. Smelter payment for both gold and silver is most satisfactory when lead concentrate is maintained at about 30% lead.

Test work on Silver Standard ore was done by Denver Ore Testing Division and Wright Engineers, Ltd., Vancouver, who designed the milling plant.

Typical feed averages per ton

GOLD	0.0635	oz.
SILVER	16.6	oz.
LEAD	1.25	%
ZINC	4.15	%
CADMIUM	0.04	%

The Result:

		Each ton Lead concentrate contains	Each ton Zinc concentrate contains
GOLD	(ounces)	1.0025	0.052
SILVER	(ounces)	366.05	17.9
LEAD	%	30.4	1.0
ZINC	%	8.2	51.5
CADMIUM	%	—	0.60
IRON	%	14.8	6.5
SILICA	%	6.2	5.8
LIME	%	1.0	0.8
ARSENIC	%	6.4	—
ANTIMONY	%	1.2	—
MOISTURE	%	8.9	12.6

Used All Over the World

Other operators also have firsthand knowledge of high metallurgical recoveries made by Denver "Sub-A's." Over 33,000 Denver "Sub-A" Flotation Cells are recovering valuable concentrates—*at high net profits*—all over the world. *Detailed information on the results you may expect from Denver "Sub-A" Flotation Machines can be determined by the Denver Equipment Company Ore Testing Laboratory.*

For Your Engineering Notebook

The complete story of high flotation recovery at Silver Standard Mines, Limited, appeared in the ENGINEERING NOTEBOOK SECTION, May-June issue, Deco Trefoil. Engineering data on successful operations is supplied as a service to the industry. Reprints of Engineering Notebook Sections reporting on operations similar to yours will be sent to you without cost. Address your request to our Publication Division

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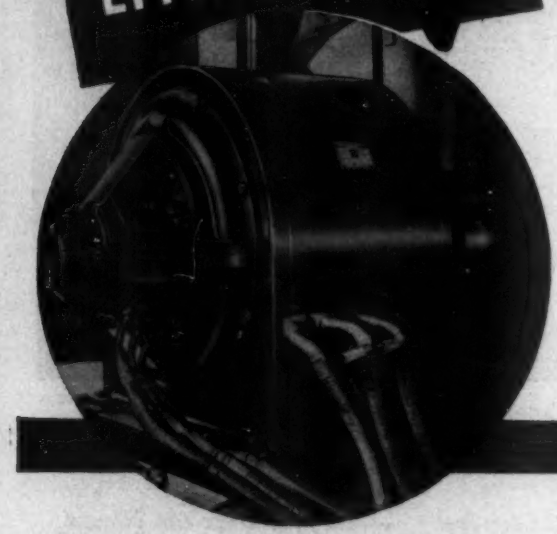
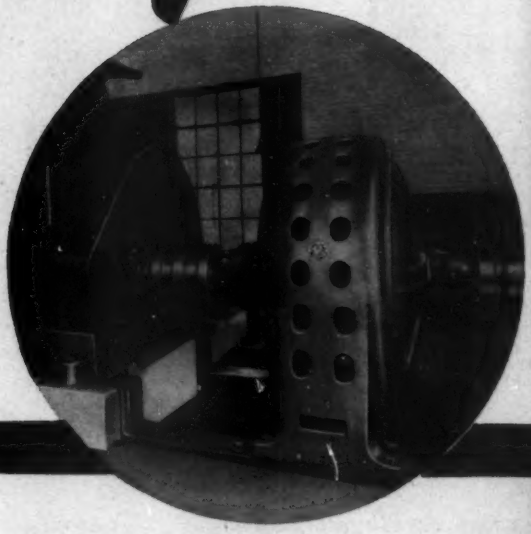
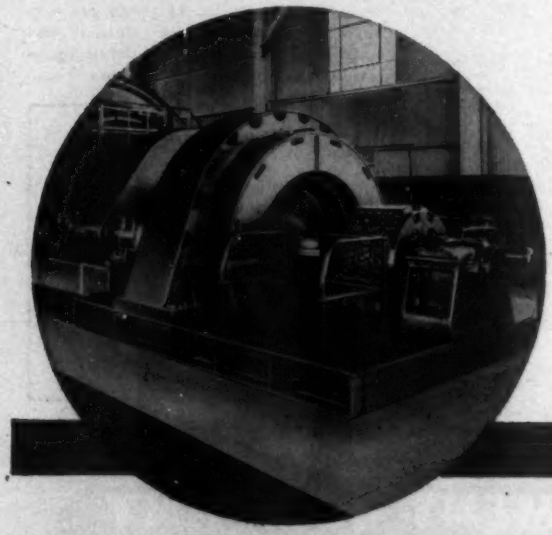


A typical G-E wound-rotor induction motor, 1000 hp, driving an ore hoist at a lead mine. G-E induction motors, ranging in capacities from 50 to 3500 hp, provide the simplest, least expensive method of obtaining the advantages of electric drive for mine hoists.



This 600-hp G-E induction motor is one of 16 identical units in a large Eastern mining operation. To meet specific duty cycles, G-E mine-hoist control uses the "definite-time" accelerating relay system that requires fewer interlocks, employs simpler panel wiring, and affords longer relay life.



A typical G-E 3500-hp wound-rotor induction motor geared to a main mine hoist. G-E motors and control equipment for mine hoists, designed and built for rugged service, have been on the job for many years in hundreds of mines throughout the country.

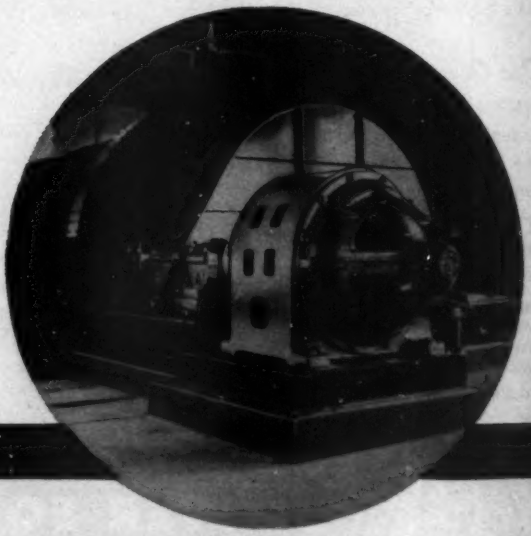


Photo shows a G-E 250-hp wound-rotor induction motor driving a man-and-material hoist in a Wyoming metal mine. G-E mine-hoist induction motors are low in upkeep cost, and require little or no maintenance attention. They need only routine lubrication and occasional brush renewal.

XUM

all three

with a G-E INDUCTION MOTOR Mine-Hoist Drive!

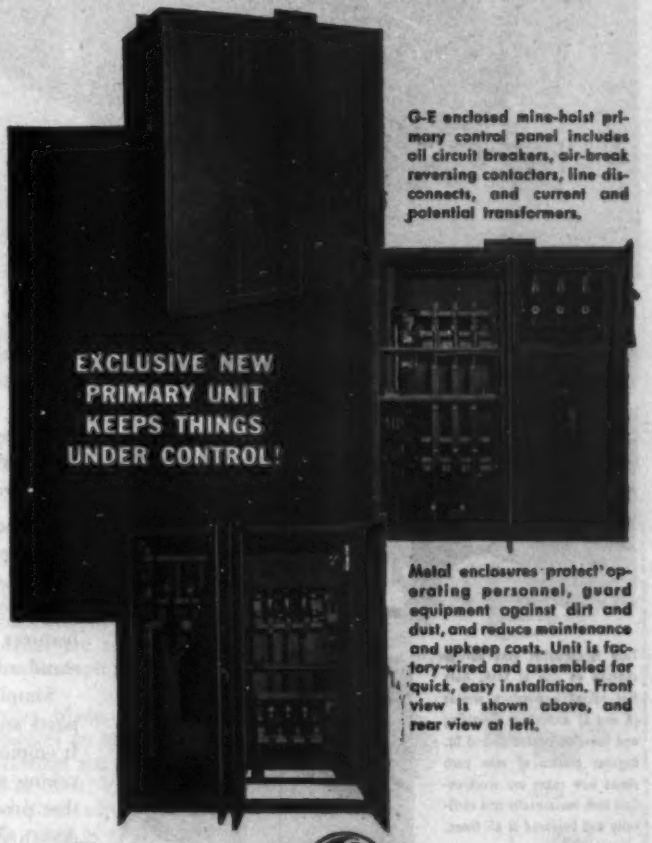
Here's your reliable, low-cost way to high-tonnage ore hoisting! With a General Electric a-c mine-hoist drive, you combine the basic advantages of simple operation, low first cost and upkeep, and high efficiency.

In the G-E wound-rotor induction motors used for this service, the stator and rotor windings—specially insulated and securely braced—are ample for full "plugging" operation. All hoist-motor rotors are also strongly banded in place to withstand frequent reversing and above-synchronism operation.

SIMPLE CONTROL. Speed regulation is simple, and controlled through primary and secondary contactors by a master switch mounted at the operator's platform. Seven to ten successive speed steps are provided, from standstill to full-speed running position. Efficient, adjustable-speed control of overhauling loads is obtainable, where indicated, by dynamic braking, in which d-c excitation is applied to the motor stator windings.

G-E hoist control is designed to give maximum protection to machines and personnel in the event of abnormal operating conditions.

More than 700 performance-proved G-E drives—both a-c and d-c—are now helping mines throughout the country to step up production, and reduce hoisting costs. Whether you're installing a new hoist or modernizing an existing one, an experienced G-E mining specialist will gladly size up your problem, and help you meet it economically. Call him at your nearest G-E office. *Apparatus Dept., General Electric Co., Schenectady 5, N. Y.*



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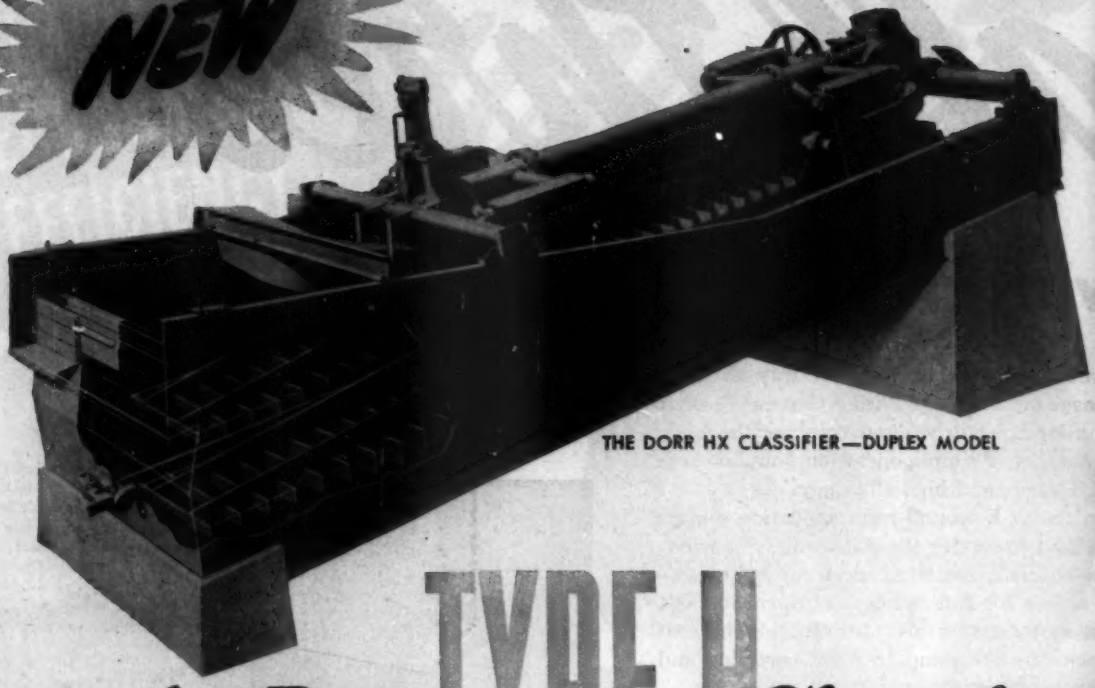

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◀ Chalk marking the rake path of a duplex Dorr HX Classifier in shop demonstration.

Type H Classifier rake path ▶ chalked on side of tank. Note flat raking and return strokes (A and C) and elliptical raising and lowering strokes (B and D). Regular pattern of rake path shows how rakes are synchronized both horizontally and vertically and balanced at all times.

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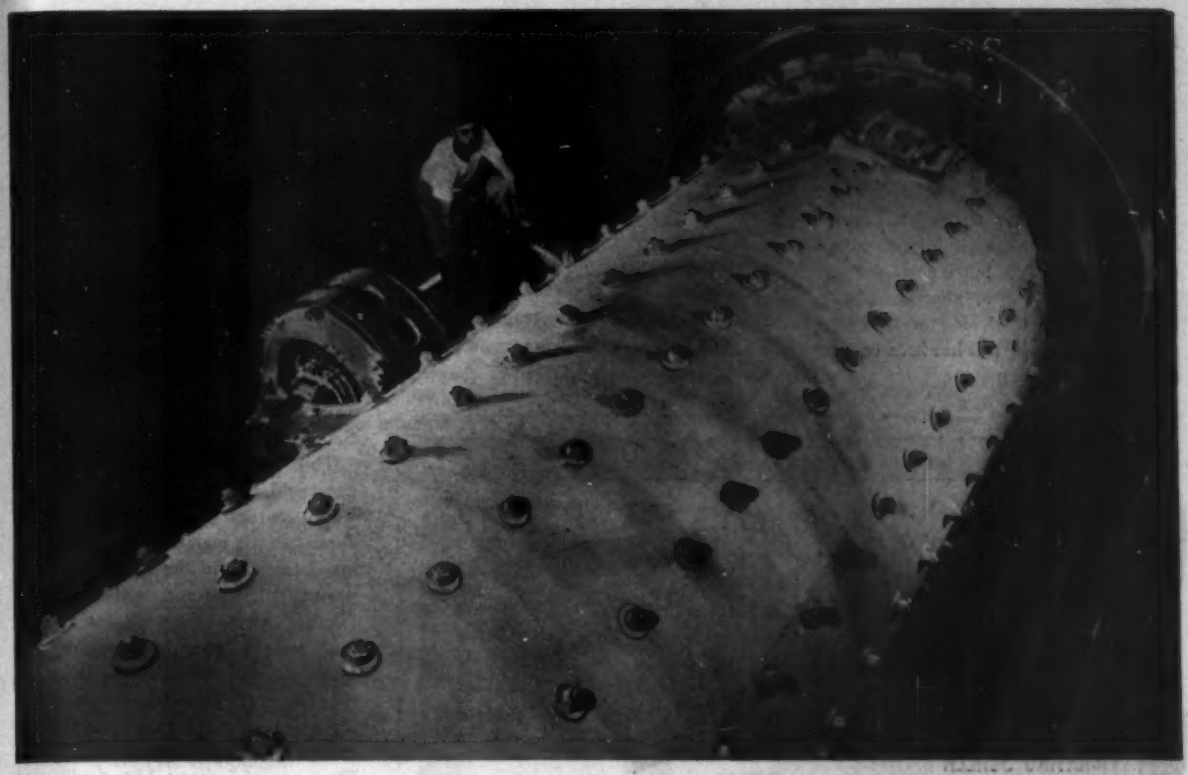

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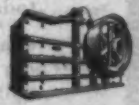

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XUM

Book Reviews

Salesmanship At Its Best

During the course of some fifteen years of close association with AIME, and especially with its publications, we have undertaken at the Institute's request a large variety of jobs, most of which involved putting down on paper one word after another in an effort to express an idea or explain a fact. Occasionally this has been easy; most of the time, however, it has been difficult, so difficult in fact that we have unquestionably been responsible for one of the world's greatest collections of still-born brain children.

But even the most difficult of these efforts and even the most copious quantities of "blood, sweat, and tears" that accompanied these struggles pale into insignificance in comparison with the job we were requested a couple of weeks ago to do (a request, by the way, that came from E. H. Robie, and which may have been inspired by the fact that, while he was in San Francisco, we had written and slipped surreptitiously into the personal column of the March issues of the AIME Journals, a short biography of the new secretary). The request was to write something about the most recent volume of the Seeley W. Mudd Series, a volume which has been given the fearsome and awe-inspiring title "Bergwerk- und Probierbüchlein" and whose authors are Anneliese Sisco (our better half) and Cyril Stanley Smith.

This volume contains the translations into English of two German sixteenth-century works, which should appeal to all members of AIME who are interested in the history of their profession. The originals are the "Bergbüchlein," the first text on mining geology ever printed in any language, and the "Probierbüchlein," the first printed work dealing with the assaying of ores and metals and thus with any aspect of metallurgy. They are of interest historically and for the character of their subject matter, and also because there is no question that the great Agricola used both of them as source material when he wrote "De Re Metallica." But they are humble works of literature compared with Biringuccio's (1540), Agricola's (1556), and Ercker's (1573) beautiful books, which followed them.

The translation as it appears in this Mudd Series volume is an excellent one and retains in large measure the flavor of the sixteenth-century originals. We are sure of this—we should be, having lived with it for years, during which time we witnessed the mental and spiritual anguish that accompanied the search for just the right English equivalent of a German mining or metallurgical term that had disappeared from the German language in the more than 400 years since it was used.

We are also certain that the English text is technically correct; Mrs. Sisco could have found no better collaborator on the "Probierbüchlein" than Cyril Stanley Smith, whose translations (with Mrs. Gnudi) of Biringuccio's "Pirotechnia," also published by the Mudd Fund, was a best seller for some years and now is a col-

(*Continued on p. 43, Section 1*)

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MINING ENGINEERING

Guarding the Public Weal

With the passage of the bill empowering the President to overhaul our sprawling federal bureaucracy, the prospect is bright for the enactment of the recommendations of the Hoover Commission. If the blueprint laid out by the Commission is followed, the mining industry stands to benefit doubly—as taxpayers and as members of that industry. As taxpayers, we will benefit from the estimated three billion dollar saving and improved government administration; as members of the mining industry our task will be lightened by the revision of mining laws as proposed by the Commission's Task Force on Natural Resources.

Fundamentally, our present system for location and patenting is sound, but mining men have long been plagued by the confusion and lapses in Land Office records, confusion in ownership from extra-lateral rights and location on the assumed course of a geologic body, withdrawal of land from the prospector's domain, and government red tape in securing patents. To eliminate these and other abuses the Hoover Commission recommends eleven revisions in the laws pertaining to mining locations and patents, and four to laws for leasing.

The steady decline in the number of claims patented, from 2500 in 1905 to less than 100 in recent years, should be a matter of grave concern to those interested in American self-sufficiency in mineral resources. With the depletion of our present mineral resources it is expedient to pass laws that encourage exploration. In formulating these laws, new methods of prospecting, such as geophysical methods, should be taken into consideration so that tracts of land of appropriate size may be opened and protected for the necessary time required to prove the existence of deeply-hidden deposits.

Now that reorganization has been approved by Congress, some eighteen major pieces of legislation must be passed before the Hoover Report can be fully implemented. Today's problem is this: major measures based on the report, while showing real strength, are in danger of becoming caught in the "log jam" of traditional legislative log-rolling. The bill approving reorganization provides that either the House or the Senate may kill a Presidential reorganization plan by adopting a resolution of disapproval. Such a veto resolution must be adopted by a "constitutional majority" of either branch—that is, by 49 Senators or 218 Representatives, not just a majority of those present. In addition, the bill provides that Congress must have sixty days of continuous session in which to make up its mind whether or not to veto a Presidential reorganization plan. Since Congress plans to adjourn before Aug. 20, the prospect is dim for any reorganization before 1950.

However, this delay permits time for public sentiment to gain sufficient impetus to impress on the mind of every legislator that we, the people, will brook no modification of our fixed intention to have efficient and economical administration of our government. Testimony to this conviction was the spontaneous formation of the Citizens Committee for Reorganization of the Executive Branch of the Government which will act as a watch dog to Congress and guardian of the public weal.

Mineralogical Methods in Mineral Exploration

By PAUL F. KERR • CHAIRMAN, DEPARTMENT OF GEOLOGY, COLUMBIA UNIVERSITY; MEMBER, AIME

The insufficiencies of our mineral resources are becoming well known, and the national political conscience seems to be troubled at last by our dependence upon mineral commodities which must come from other lands. This realization places increased responsibility upon the industrial, educational, and governmental groups whose task it will be to solve the problems of mineral supply. The assignment comes at a time, however, when the methods of mineral search themselves constitute a problem for study.

As we all know, from the time that man first sought mineral deposits until recent years, prospectors have been largely responsible for mineral discoveries. Today the era of the prospector is almost over. The human ants that have literally examined all accessible outcrops and have tested the gravels of nearly all stream beds in the United States have left undiscovered few worthwhile mineral deposits within the scope of their recognition. The pages in the history of mineral exploration are now being turned from the era of the surface prospect to the era of the hidden deposit.

Fig. 1. Fine-grained thermally reactive materials may be identified by contrasting their differential temperature curves with that of an inert material (aluminum oxide). The four units shown above for the differential thermal analysis process are from left to right: furnace for single samples, furnace for multiple samples, furnace control recorder, and differential recorder.

Hidden ore bodies may be found and are being found chiefly by scientific methods. Geophysics has received much attention, and varying degrees of success have been reported through the use of magnetic methods, different modifications of electrical methods, gamma-ray counters, seismic methods, and even gravimeters. But it seems likely that credit for most recent discoveries should go to a combination of good geology plus drilling. Likewise, it seems to be demonstrated that no one adequate method of universal application has yet been found or is likely to be found. Rather the solution of the problems of mineral exploration appears to rest upon the adaptation of suitable techniques to the area of immediate interest. Further, the time has arrived when geology, which remains the cornerstone of mineral exploration, must include more than the time-honored conventional methods of field

mapping. The co-ordination of field and laboratory methods of study as long advocated by a number now appears closer to being realized.

mineralogy in mineral exploration

Events indicate that laboratory methods identified with mineralogy are being used increasingly as a supplement to geological work. These methods are finding their way into field application with the advent of co-ordinated programs of mineral exploration research.

One phase of such mineralogical research is concerned with the development of suitable methods for studying the masses of weathered, oxidized, and altered materials commonly found in the proximity of mineral deposits. The relationship between metallic mineralization and wall-rock alteration constitutes a major problem. The significance of the texture, color, and composition of gossan is another feature worthy of further study. Roland Blanchard and others, who have studied leached outcrops, have contributed criteria of much value. Yet this is a field in which further contributions seem destined to be made.

Mineralogical methods, themselves, are not new in principle any more than the methods of geophysics establish new physical principles. Differential thermal analysis which is now being employed in the study of the altered materials around ore bodies, in the examination of limestones and other carbonate masses, in the estimation of the nature and amount of clay material in weathered or altered rocks, and in a number of other ways, is in fact, the reapplication of a method which has long been known. The same may be said for microscopic, X-ray, spectrographic, and other techniques.

However, there has been a constant improvement in the knowledge of the application of each of these methods, in their co-ordination with each other and in their application to the general problem of mineral exploration. The cumulative effect may not appear spectacular but the ultimate results from the detailed application of such methods seem certain to be considerable.

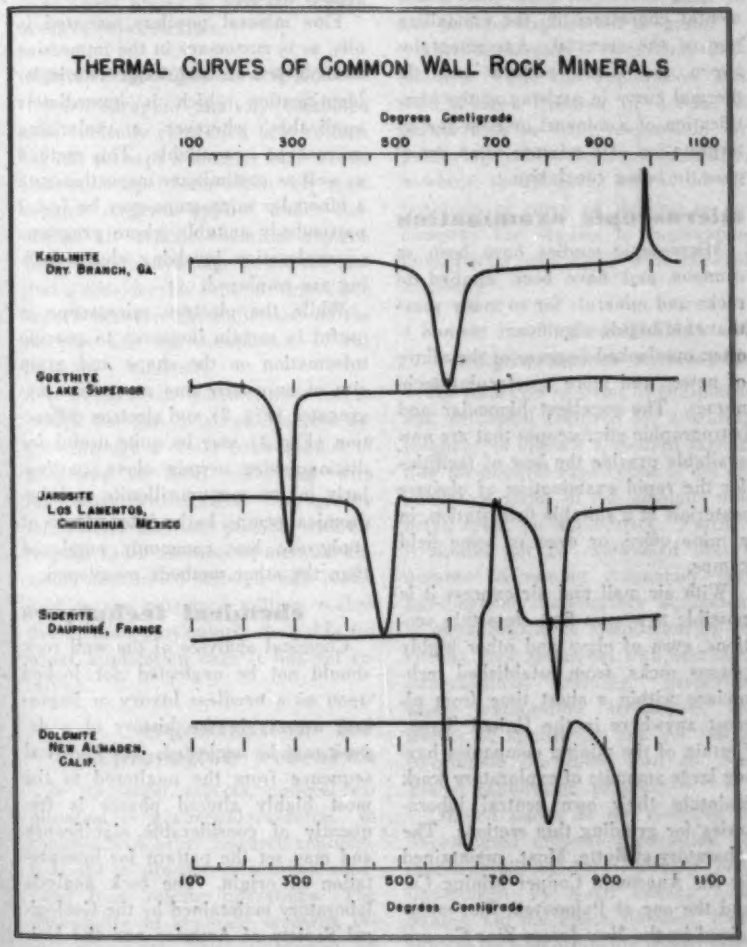


Fig. 2. Representative differential thermal curves of several minerals showing both exothermal (upward) and endothermal (downward) reactions between 100 and 1000C.

field methods

At the risk of being considered pedantic, it is only fair to say that one of the most important items of equipment in all mineralogical research, whether in field or in laboratory, is a good hand lens. Obviously, it is not the lens itself which is so important. The background of useful knowledge concerning minerals and rocks which is acquired through systematic study plus the constant and efficient use of this simple aid is essential in any program of mineral exploration.

It should also be kept in mind that the mappable units in any particular area are largely determined by the amount of discrimination feasible in the field. Close visual examination with the advantage of laboratory experience often determines the number of units suitable for mapping in a given area.

One section in which mineralogical methods are finding application lies in the Cordilleran region of the western United States. Throughout this region attention has long been directed to mineral deposits associated in one way or another with the many areas of igneous intrusion. Long ago, B. S. Butler, Waldemar Lindgren, James F. Kemp, and other keen students recognized the important role that igneous processes and their aftermath have played in the original emplacement of metalliferous deposits throughout the area.

More recently, Reno Sales, T. S. Lovering, C. M. Schwartz, and others have been devoting considerable attention to the nature of the processes which have brought about wall-rock alteration as an accompaniment of ore deposition. Such investigation, which may be referred to as an alteration

study, involves the use of a variety of techniques.

differential thermal analysis

One of the most useful of the laboratory methods (Fig. 1) which may be utilized to accompany field mapping is differential thermal analysis. Differential thermal curves (Fig. 2) not only yield the thermal history of a mineral, usually from 0 to 1100C, but furnish a curve which may be compared with standard curves as a basis for identification.

The method is particularly desirable for the examination of fine-grained materials devoid of the simple characteristics by which large and well-formed crystals of individual minerals may be recognized. It likewise has considerable application in the examination of fine-grained mixtures, where a mass consisting of two or more clay minerals or other fine-mineral aggregates requires resolution into its mineral constituents.

The following list includes a number of common minerals or mineral groups often found in association with mineral deposits in minute aggregates, but which are reasonably amenable to the method of differential thermal analysis: clay minerals, carbonates, zeolites, serpentine group, limonite group, manganese oxides, jarosite group, chlorite group, alunite group, quartz, phosphates, rare earths, bauxite group, vermiculite.

Individual members of these mineral groups may be identified, and, frequently, an approximate quantitative estimate of the relative proportions may be made, even in fine aggregates. The extent of the information that may be developed in an objective manner makes this a most useful laboratory auxiliary.

X-ray diffraction

About 29 years ago X-ray diffraction was first applied to mineral identification. Since that time dozens of laboratories throughout the country have installed diffraction equipment. When differential thermal analysis is accompanied by X-ray diffraction study, each method tends to confirm the other and together the two provide much useful and reliable information.

It is now routine practice to mount a mineral sample weighing a few milligrams and ground to —325 mesh, pass a beam of X-rays through the sample and collect the pattern of the diffracted X-rays on a narrow strip of photographic film. Some equipment on the market even records the pattern automatically on a roll of paper.

Courtesy of American Cyanamid Co.

Fig. 3. Shape and grain size of unusually fine crystalline aggregates can be determined with the electron microscope as shown in this electron micrograph of kaolinite crystals which are magnified x 35,700, reduced about 1/3 in reproduction.

Diffraction patterns provide a form of comparison based upon that fundamental characteristic, the crystallization of the material. Agreement between the X-ray pattern and the thermal curve in arriving at the identification of a mineral or even the interpretation of a mixture often comes close to being conclusive.

microscopic examination

Microscopic studies have been so common and have been applied to rocks and minerals for so many years that this highly significant method is often overlooked because of the allure of newer and more spectacular techniques. The excellent binocular and petrographic microscopes that are now available provide the best of facilities for the rapid examination of obscure materials at a suitable field station, in a mine office, or even in some field camps.

With air mail and air express it is possible to secure first class thin sections, even of clays and other highly porous rocks, from established technicians within a short time from almost anywhere in the United States. Certain of the mining companies having large amounts of exploratory work maintain their own central laboratories for grinding thin sections. The laboratory at Butte, Mont., maintained by the Anaconda Copper Mining Co. and the one at Palmerton, Pa., maintained by the New Jersey Zinc Co. are examples. The work of Charles Meyers at Butte, in developing apparatus for sawing rock slices almost to thin section thinness has constituted a considerable contribution to the technique of section grinding. The facility with which sections may be cut in this way, together with the unusual size of the section area are both worthy of note.

It seems likely that today, as in the past, the chief contribution to come from microscopic work lies in the interpretation of the history of the rock under study. When the precision in identification attainable through the use of differential thermal analysis and X-ray diffraction are combined with the textural criteria observable with the microscope, a considerable contribution to knowledge of the alteration of the wall rock becomes feasible. This provides a basis for examining such fine aggregates as may be formed by clays, sericite, alunite, and jarosite in a number of the western copper deposits.

Fine mineral powders mounted in oils, as is customary in the immersion method, provide additional criteria for identification which is immediately applicable wherever a polarizing microscope is available. This method as well as preliminary inspection with a binocular microscope may be found particularly suitable where programs of exploration involving churn drilling are employed.

While the electron microscope is useful in certain instances to provide information on the shape and grain size of unusually fine crystalline aggregates (Fig. 3) and electron diffraction (Fig. 4) may be quite useful for distinguishing certain clays, particularly in the montmorillonite and hydromica groups, both of these types of study are less commonly employed than the other methods mentioned.

chemical techniques

Chemical analyses of the wall rock should not be neglected nor looked upon as a needless luxury or important aspects in the history of a deposit may be neglected. The chemical sequence from the unaltered to the most highly altered phases is frequently of considerable significance and may set the pattern for interpretation of origin. The rock analysis laboratory maintained by the Geological Society of America and the University of Minnesota, or several commercial analytical laboratories may be relied upon for analytical work.

Some advocate the use of staining tests for the clay minerals and related species. Reagents such as congo red, malachite green, etc., will produce stains in some clays. The advantage of staining tests lies in their rapid application in the field. The disadvantage lies in the fact that the same dyes will not always react in the same way with the same mineral. Further, the influence of the wide variety of impurities with which clay minerals may be associated has not been adequately determined for the various stains. If these obstacles can be satisfactorily overcome, the use of stains should be of great assistance in the mapping of clay-bearing areas.

Spot tests for various elements and certain sensitive procedures for the detection of several of the metals offer the possibility of widespread application in field mapping. A careful map of the distribution of traces of elements may indicate the loci of the most likely places to look for hidden areas of concentration.

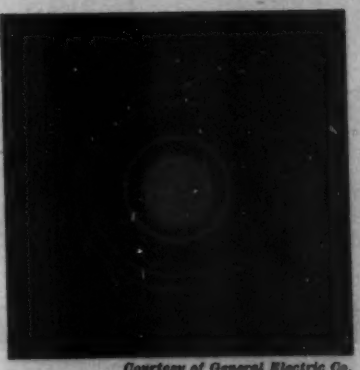
spectrographic analyses

Spectrographic analysis combined with carefully controlled field collecting has perhaps a sounder basis chemically than spot tests which rely upon organic reagents applied under conditions of variable chemical association. The disadvantage lies in the fact that a considerable lapse of time often occurs between the collection of the samples and the completion of the laboratory studies.

It would seem feasible, however, to bridge the gap to a considerable degree through a closer co-ordination of programs of field collecting with laboratory investigation. The quantitative determinations possible with extremely small amounts of metallic elements distributed through wall rocks, even to one part in a million, makes this method particularly desirable for wider application than it has yet received.

organization for exploration research

In its larger aspects, mineral exploration is geological research. In common with other research ventures, it is apt to prosper most when separated from routine operations. The desirability of separate research organizations has been recognized by major manufacturing concerns but for the most part it seems to have been overlooked by the mineral industry. Too frequently exploration departments, even where they exist and are well staffed, are forced to bow to the whims of operating executives instead of having a free hand to apply tests and develop principles. Freedom of action is a most important asset in any mineral exploration program.



Courtesy of General Electric Co.

Fig. 4. Electron diffraction analysis is useful in determining clay minerals.

Another essential is research responsibility. While the exploration unit should have freedom of action, it should also be held responsible for producing results. Such responsibility, however, should carry with it the opportunity to carry on drilling to accompany the studies in exploration immediately concerned with mineral discovery.

feasibility of exploration research

The total cost in terms of equipment and personnel required for a mining company to operate a modern laboratory for mineral exploration is not excessive in terms of the amounts now being spent for exploration. Further, it should not be overlooked that a number of mining companies may have entirely satisfactory equipment in metallurgical or manufacturing divisions, good geologists with first class mineralogical training, properties where exploratory techniques are worthy of application and yet fail to co-ordinate the three into an integrated exploration program.

Limited moves in the direction of co-ordinated mineral exploration research programs have already been undertaken by some mining companies. On the whole, however, these companies have probably shown much less of a tendency to undertake such research programs than would be the case with manufacturing concerns of comparable magnitude.

In some quarters the impression prevails that all research in mineral exploration should be left to government agencies or to universities. This point of view fails to take into consideration the purposes for which the government agencies and the universities exist. Presumably it is the object of a mining company to find, develop, and mine a mineral deposit at a profit. Even mineral exploration research is only a means to an end. Since mining companies are private enterprises their efforts are apt to be largely competitive.

Government agencies, on the other hand, have a responsibility to serve the public. This means that research developments, where made by such agencies, must become public property and should not be too closely focused for the benefit of a particular mining company. The fields in which government may be most effective are in fundamental research or even more in the direction of public surveys. One of the projects in the realm of government which should be of inestimable public benefit would be a complete detailed geological map of the entire mineralized area of the States.

The role of the universities lies in the field of education and research. If both industry and government are to have the highly trained personnel necessary to carry on mineral exploration research to the extent that the situation requires, greater care must be exercised in the selection and training of graduate students. Staff members must also be encouraged and aided in exercising that initiative in the development of improved techniques which it is hoped will ultimately lead to new discoveries.

If one is to picture the mineral exploration research program of the future, it is a vision of the intensive application of those field and laboratory methods which have been demonstrated to have particular merit in individual areas. To this pattern government should contribute more detailed and more extensive topographic and geologic maps, the universities should supply new personnel and studies in fundamental science, and the companies, themselves, should be constantly on the alert to develop and apply new techniques advantageously.

XUM

Open Pit Forum

continuing a new department which is intended to report the latest practices in open-pit mining. We want your experience and discussion

Truck Body Cleaning

BY C. A. LINDBERG • OLIVER IRON MINING CO.

Several new methods have been developed on the Iron Range to remove the material adhering to truck bodies in freezing weather. A machine known as a Gradall, incorporating the features of digging both above and below the horizontal, digging angle of blade control, full revolving, travel almost anywhere, and operated by one man, has proven successful in trials at body cleaning. Several of these units are in use with great saving in box-cleaning costs and improvements in efficiency of the truck operation with full capacity boxes. In addition to their winter use, the same machines have been used for ditching and excavating in the summer with great saving.

The latest method of combatting this freezing problem is to carry the exhaust from the engine through the body, making use of the tremendous heat that is now being wasted. To do this, the present wood sandwich which is used between the bottom of the box and the wear plate is removed and replaced with angles or channels run lengthwise of the box and spaced to permit gases to flow between or through them and also provide a good support for the wear plate. The channels or angles are cut a little short on each end to form a cross passage that permits the gases to flow through the area. In the front end, openings are made in the wear plate and connect to a manifold created by welding a plate across the whole front of the box, spaced from the original end by the original vertical reinforcing angles at that point. The body being described is for a Euclid truck which is the only truck that has been adapted to this body-heating idea to date. At the rear of the truck, the holes from the cross passage are made in the bottom plate of the body and lead into a triangular-shaped body cross member under the scoop end of the truck.

The heating effect is remarkable. On two trial installations on the west end of the Range last winter there was never any necessity for cleaning the bodies of these trucks. The company experimenting with this method is converting all of their trucks to heated bodies this year. It appears to be the complete answer to winter body-freezing problem. It is also an excellent answer to the exhaust-muffler problem.

Patent rights for the installation have been assigned to one of the leading heavy-duty truck manufacturers who will charge a royalty fee of $50 per truck for any heated body installations made at the factory of any truck manufacturer. To expedite the addition of this heated-body idea to trucks in the field, some operators are purchasing their trucks less the wood sandwich and with the wear plates not welded to the body.

Condensed from a paper on auxiliary equipment in open-pit mining presented at the University of Minnesota Annual Symposium at Duluth, Jan., 1949.

Blasting at Cananea

Blasting at the open-pit of The Cananea Consolidated Copper Co., Sonora, Mexico, is in a monzonite porphyry rock which, in general, might be classed as medium-hard; however, there are sections which are more highly silicified than the average and are consequently much harder. The mining calculations are based on a factor of 12.5 cu ft per ton for the monzonite porphyry, and 11.5 cu ft per ton for the more intensely silicified areas.

primary blastholes

Primary blastholes are drilled by 42T Bucyrus-Erie churn drills having a 4200-lb stem, 32 ft in length and using a 9-in. diameter bit. For the past month 117 ft per operating shift were drilled; 67 ft. per bit. In the harder silicified areas 83 ft per operating shift were drilled, 52 ft per bit. Ten-inch casing pipe is used to case each hole, lengths of casing depending upon the amount of loose fill to be passed through before reaching the solid. After

This paper was prepared for the Arizona Section, AIME, meeting held at Tucson, Ariz., Dec. 7, 1948.

blasting, the casing is salvaged and returned to the churn-drill bit shop where the dented sections are forced into shape by a hydraulic jack and the torn sections are cut off with an oxyacetylene cutting torch. The short pieces of casing thus developed are butt welded together into the desired lengths and returned to service.

The majority of the blasts are in the medium-hard to hard areas; however, there are sections of the overburden close to the surface in the west pit where the rock is rather well fractured, not highly silicified, and relatively easy to break. Two types of explosives manufactured by the Apache Powder Co. of Benson, Ariz., are used; these are 8 by 18-in. 60% Special Gelatin cartridges, and LDX 705, a free-running dynamite. The 60% Special Gelatin is used in the harder overburden and in the ore, and the LDX 705 is used in the near-surface overburden blasts in conjunction with the gelatin.

toe holes and bench height

The toe distance and the spacing between the churn-drill holes have changed for two reasons since last reported at the Open-Pit Section meeting held at Castle Dome, Ariz., March, 1947. First, a rock more intensely silicified and much harder was encountered and second, the height of the bench mined was increased from 40 ft to 60 ft.

While still mining 40-ft benches, a section of the more intensely silicified rock was drilled and blasted, deck loading the 9-in. diameter blastholes which were 25 ft back of the toe and on 15-ft spacing. This close spacing between holes resulted in an extremely blocky condition so that it appeared that the ground was sheared off en masse and shoved foreward. To offset this condition the toe distance was reduced from 25 to 18 ft and the spacing between holes was increased from 15 to 20 ft, decreasing the burden on the individual hole but slightly. Using the same powder factor, this change in blasthole location was very successful for the action of the blasting forces now tended to break through the mass thus producing fragmentation rather than merely exerting a push against the entire block.

At present, in mining benches 60-ft high, the blastholes in the average hard-ore areas are drilled with a 20-ft toe hole on 20-ft spacing. In the extremely hard areas, the blastholes are drilled with a 17-ft toe and a 23-ft spacing. Deck loading in the average hard-ore areas normally consists of 7 cases of 60% gelatin in the bottom, 1½ cases of gelatin at 35 ft from the collar of the hole, and 1½ cases of 60% gelatin at 25 ft from the collar. In the extremely hard-ore areas, after recent experimentation, relatively good fragmentation was obtained loading 8½ cases of 60% gelatin in the bottom of the hole and placing 4½ cases of 60% gelatin at 30 ft from the collar. The throw of the rock using the single-deck charge is about the same as with two deck charges, the ratio of bench height to throw being 2 to 1 with a few fragments going out as far as 3 to 1.

The deck charge load was determined by calculating (1) that the powder placed below grade (about 2¾ cases) would be sufficient to break up the bottom; (2) that the powder above bench elevation, at a factor of about 3.5 tons per pound of powder, would break half of the bank or 30 ft; and (3) that the powder in the deck charge, at a factor of 4 tons per pound of powder, would break the upper 30 ft of bank. Using this loading method 3.6 tons per pound of powder were broken and 37 tons per foot of hole blasted. The bank breaks clean and requires no scaling. The tons loaded per shovel-shift show an increase of 10% over a similar blast having more than one deck charge.

Six Ingersoll-Rand 35-lb jackhammers equipped with ⅞-in. quarter octagon steel in 2½ and 4-ft lengths using 1⅜-in. Liddicoat bits are in service for blockholing boulders and for drilling high bottoms. Primarily as a safety measure to minimize falling over the boulders due, in part, to the heavier equipment, these light jackhammers supplant the heavier 50-lb hammers. The accident rate has been materially reduced. The use of the Liddicoat bit has increased the footage drilled per shift. Dynamite used for secondary blasting is 1⅛ by 8 in. 40% gelatin. One Gardner-Denver WBK 500 portable compressor powered by a D 13000 Caterpillar Diesel engine is used to permit a more flexible secondary drilling operation.

What Graduates Expect of the Coal Industry

By WILLIAM N. POUNDSTONE • 1949 GRADUATE WEST VIRGINIA UNIVERSITY

What attracts young engineering graduates into the coal industry? What do these young men expect of a career in coal mining? These questions are often asked and debated by mining men throughout the country. In order to present a comprehensive answer to them, an opinion survey was conducted at West Virginia University's School of Mines among the junior and senior classes and part of the sophomore class. The tabulation of the answers showed that the present-day mining engineering student is an older, more mature individual as compared with former years and as compared with other present-day college students.

The average age of the country's record-breaking 150,000 college graduates is 24 years. Among this year's mining graduates at West Virginia University, 78% are of this age or older. Over half (54%) of those polled in the survey are married. This is compared to 30% as a national figure. Of those married, 38% have one child and over 6% have two children. The entire senior group covered by this survey has averaged almost three years in the armed forces. An article entitled "The Class of '49" in the June issue of *Fortune* magazine states that this year's class is the oldest, most mature class in history. Our survey indicates that the mining engineering students at West Virginia University are well above the average in this respect.

A unique feature of the present-day mining student is the amount of working experience he has in his field. About 96% of this year's graduates have actual mining experience. The average for this group was 21.4 months with over 17% having five years' mining experience. This means that approximately one out of every five graduates is eligible to take his mine foreman's examination upon graduation. The experience record of the next two years' graduating classes is not quite so high; however, many of these students will work between now and their graduation during the summer months and, in some cases, at night while attending school. Next year's graduating class shows 86% with mining experience averaging over nineteen months and 12.5% with over five years. The following year shows 92% with experience averaging 9.4 months.

One of the questions asked in the survey was "What general type of work do you plan to enter eventually?" Almost two-thirds, or 63.5% of those questioned, expected to enter production, while 30.2% preferred engineering, 4.7% said "sales" and only 1.6% planned to specialize in coal preparation. These figures seem to bear out a modern trend of engineering graduates into fields that are not strictly engineering. However, college training lends itself equally well to engineering and mine management.

As a matter of interest the survey asks the questions "What starting salary do you honestly think you should receive?" and "What starting salary do you think you will receive?" The seniors thought they were worth $358 per month but expected to receive $347. While these figures are

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rather high as compared to the average starting salary for all engineering graduates, in many cases, the answer is based upon actual salary offers received by the students.

As is seen in the following tabulation, over 20% of those polled expect to step into jobs as section bosses and mine foremen.

Expected First Job After Graduation

Job	*Percent*
Surveying crew	37.5
Section boss	12.5
Mine foreman or assistant	10.7
Labor at face	32.1
Tipple	3.6
Draftsman	3.6

It is noted that the number of men expecting to go directly into engineering jobs (surveying crew men and draftsmen) totals over 40%. Since this figure is higher than the 30% desiring engineering positions as an ultimate goal, it is concluded that some of the men working toward production jobs wish to get some engineering experience. This is supported by the overwhelmingly affirmative answer given to the question "Are you in favor of the organized training program as first employment?" Only one man of the 63 polled said he was not in favor of this type of training. This would indicate that the graduate of today is desirous of some further training in each of the various phases of coal mining. The reason for wanting a varied work experience may be the fact that many of this group hope eventually to work into top management jobs where knowledge of the full range of mining operations will be invaluable. In reply to a question of what position they expected to attain by the age of fifty, 57.1% expect to be superintendents, 15.9% top executives, 14.3% chief engineers, 6.3% operators, 4.8% chief of sales, and 1.6% consulting engineers.

The group was also asked what salary they expected to be earning at the age of fifty. The average for the answers was $10,420 per year. This is somewhat larger than the $7,300 per year given by the Bureau of Labor Statistics as average for graduate mining and metallurgical engineers at that age. However, it is believed that salaries are somewhat higher for coal mining engineering. This figure is conservative as compared to the answers to a similar question asked a group of Tau Beta Pi men at their convention this year. Seven percent of this group expected to be making over $100,000 a year at the age of fifty, while $25,000 a year was the highest estimated salary recorded in our survey.

Another question this survey has attempted to answer is "Why do young engineers enter the coal mining industry?" The following tabulation shows the answers given:

Reasons for Entering the Coal Industry

	Percent
Offers the best opportunity	51.9
Interesting work	26.0
Raised in coal mining town	18.2
Much room for technical improvement	1.3
Previous mining experience	1.3
To improve miners' welfare	1.3

The questionnaire asked "What, if any, disadvantages do you associate with the coal industry?" The answers received were as follows:

Disadvantages Associated With Coal Mining

	Percent
None	19.2
Poor living conditions	26.5
Dangerous work	11.8
Much experience needed for good jobs	4.4
Adverse public opinion	4.4
Disturbed labor conditions	13.2
Poor working conditions	10.3
Limited social and educational opportunities	5.8
Uncertain market condition	4.4

In an attempt to find out the graduate's reactions to working in remote areas, the following was asked: "In terms of dollars per month, evaluate the difference in salary you deem necessary to offset the disadvantage of living in an out-of-the-way mine camp, say fifty miles from a large town." Seventy-six percent of those questioned felt they would have to have an average of $75 more per month to interest them in working in such a location. This would indicate that most students would like jobs near the cities. However, the answers given to the question "In what general locality would you like to work?" seem to contradict this belief. There were almost as many localities named as there were students questioned and many listed sparsely populated areas.

In order to find out what the graduate considers important in selecting a job, the questionnaire asked the students to arrange, in order of importance, nine points commonly considered in selecting a job. They are in order of importance: (1) chances for advancement, (2) salaries, (3) working conditions, (4) security, (5) housing, (6) educational opportunities for children, (7) retirement benefits, (8) social contacts, (9) location with respect to cities.

Several things are evident from the order in which the points were placed. The fact that "location with respect to cities" was placed last indicates that the men are quite willing to work in remote areas; however, as the previous question indicated they expect to be paid a higher salary to do so. This same condition exists with respect to working conditions but in this case the graduates apparently give this point more serious thought. Another point brought out is the fact that this group of men consider job security very important. While this item ranked fourth it received the second highest number of votes as the most important item.

Probably the most important conclusion that can be drawn from this survey is that the trends shown here are similar to those of this year's graduating class all over the country. They, as do the college graduates from other parts of the country, look towards security in a job. These men for the most part were born during the 1920's and have suffered through a World War and one depression. They look now towards a job that will offer some security in the event of a recurrence of hard times. For this reason many of this year's class are turning to the larger, more established companies. These men are entering the coal industry because they like the type of work and feel it offers the best reward for their technical skill and ability even though they associate many disadvantages with it. They are willing to accept these disadvantages in return for a chance at top-level management positions.

These men are equipped with a college education, but in many respects they are different from other college men. They are willing to go down into the earth and work hard under hazardous and unpleasant conditions at jobs that require no educational background, but in return they hope to win a position of respect and responsibility in the coal industry.

Trackless Mining Proposed

By H. C. LIVINGSTON • VICE-PRESIDENT, THE UNION PACIFIC

At the Hanna No. 4-A mine of The Union Pacific Coal Co. a new system of trackless mining is being utilized to extract a 26-ft coal seam. By using shuttle cars and a conveyor belt for haulage in the rooms and panel entries instead of pit cars and panel hoist a saving of 12 cents per ton is expected.

Mining is conducted in a 26-ft coal seam laid down on a synclinal structure with active workings on a pitch of approximately ten to twelve degrees. The coal seam is interbedded between a shale bottom and sandstone top. The coal is friable, high in moisture, and subject to spontaneous combustion. The mine is rated gassy by the State Coal Mine Inspection Department.

Mining is conducted at the present time by unbalanced rope haulage on main slopes spaced on approximate 4300-ft centers with the three-entry system driven to boundary lines, strike entries being turned on approximate 1300-ft centers. Panels are driven to the rise between entries at approximately 750-ft intervals with panel rooms driven on the strike 500 to 600 ft long, leaving 150 ft of barrier pillar between each block of panel rooms.

The panel between entries consists of two places on 60-ft centers driven to the rise by use of two shaking conveyors equipped with duckbills. It is necessary to install a development hoist and lay track in one place as the shaking conveyors advance up the pitch.

The haulage on panels is by panel hoist with track laid in each room and Pioneer mining of the rooms executed with track-mounted cutting machines, hand-held drills, and 11 BU Joy loaders. The top coal in rooms is extracted in two benches, the top coal being shot down in blocks 100 ft long.

Generally speaking, the present system employs track-mounted equipment with Joys loading into four-ton steel pit cars with rope and motor haulage.

proposed trackless system

In changing to the new system of trackless mining it is proposed to maintain the distance between entries of approximately 1300 ft and to drive three-place panel development to the rise by use of three shaking conveyors equipped with automatic duckbills, panels spaced on 1300-ft centers, the three shaking conveyors discharging onto a portable belt conveyor located in the center place as shown on Plate 1. You will note that the shaking conveyor driving the center place turns the panel crosscuts, with the exception of blind crosscuts, and that the shaking conveyors driving the outside places turn the room necks. A fourth place is driven at the bottom of the panel to establish positive ventilation during the period of panel development. The 26-ft coal seam is of definite advantage in providing over and undercasts for ventilation purposes.

You will note from Plate 2 that the room necks are staggered on the opposite side of the panel, with rooms centered 75 ft apart, and that the shaking conveyors are advanced every 225 ft by the addition of portable belt conveyor as the shaking conveyors advance.

It has been our experience that the small details of mine work are the important details. There is a mistake, deliberately shown on Plate 2, in procedure to point out the importance of a small work detail. You will observe that the blind crosscuts which are cut

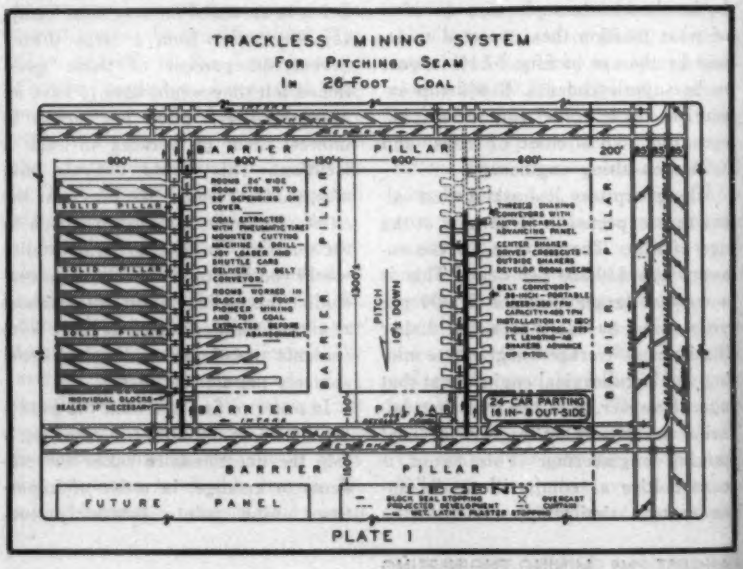


PLATE 1

This paper was presented by the Coal Division at the 1949 AIME Annual Meeting.

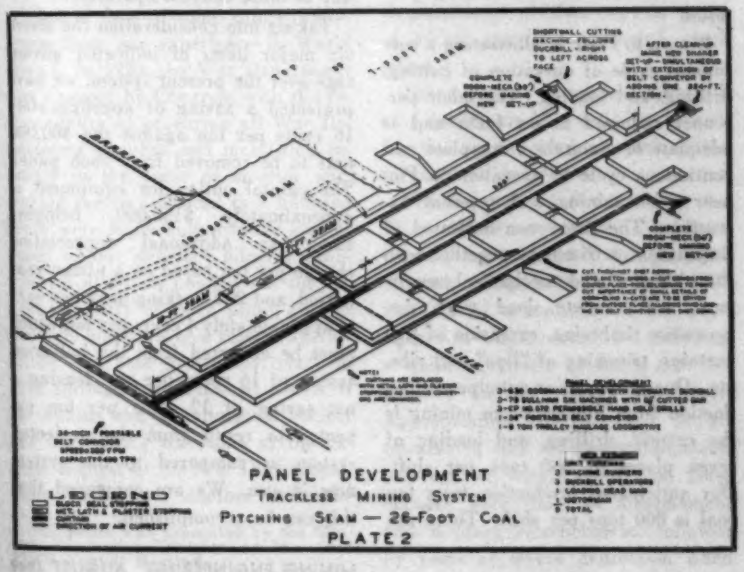
for Pitching Coal Seam

COAL CO., ROCK SPRINGS, WYO.; MEMBER, AIME

through and not shot down are shown driven by the shaking conveyor in the center place. At the time of opening this would necessitate double or triple handling of this fall of coal to the belt conveyor. The proper method is to develop the blind crosscuts with the outside shakers, leaving the fall to be shot down adjacent to the belt conveyor in the center place for ease of loading.

It is also necessary to plan the number of panel crosscuts and room necks to be developed by each shaking conveyor in order that proper balance of work is provided for each unit to keep the shaking conveyors in step in advancing the panel. This panel development system provides for a continuous cycle of operation. The cutting machine is sumped in the corner of the working place after clean-up by the duckbill and follows the loading across the working face; drilling, shooting, and loading follow in sequence. You will note that we have indicated labor personnel required, a total of nine being adequate for this operation. We anticipate that it will take 200 machine-shifts to develop the panel proper, including 20 shifts provided for moving shaking conveyors and installing the belt conveyor. The anticipated machine-shift tonnage is 67 or 200 tons per shift from the panel development operation.

The proposed method of panel development provides for the elimination of all track work on the panel, elimination of development hoist at the lower end of the panel, together with the services of a hoisting engineer and rope rider to service the shaking conveyors as they are advanced up the pitch; the belt conveyor replaces the development hoist and track, and one loading head man executes the work formerly performed by the hoisting engineer and rope rider. This setup will also improve pit-car service to the shaking conveyors and will eliminate, to a great extent, delays awaiting car service.



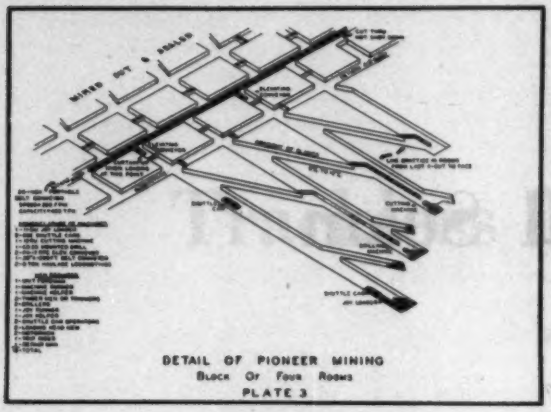
Upon completion of the panel development, the shaking conveyors are removed through the entry above, the belt conveyor is completely installed, ventilation is established to the return aircourse above, trackless mining equipment is installed, and the panel is now ready for Pioneer room mining.

Pioneer room mining in the 26-ft seam is to be accomplished by use of a 10 RU pneumatic-mounted cutting machine with places driven 24-ft wide, top cut 10 ft high, and center sheared. Drilling is to be done by CD-25 rubber-tired mounted drill; loading by 11 BU Joy loader into two 60E 10-ton shuttle cars tramming to elevating conveyor discharging onto a 36-in. belt.

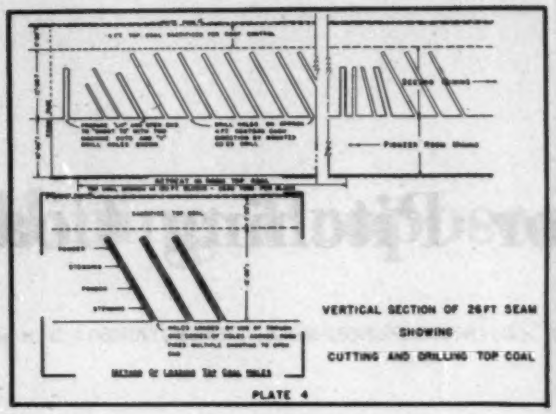
It is planned to drive slant crosscuts between rooms on a gradient of nine to twelve percent in favor of loaded shuttle cars. We plan to employ two elevating conveyors providing two points of loading onto the belt for each block of four rooms as shown on Plate 3.

During Pioneer room mining, air is conducted to the working faces by use of line brattice from the last open crosscut, as shown, with return air from the last room to the rise coursing through the panel to the return aircourse above. It is important to maintain a positive method of ventilation in a potentially gassy condition.

The major problems in top coal extraction in the Hanna field are roof control, the preparation of "lips," and top-coal drilling. This has been accomplished in the past by employing hand-held drills and an excessive



DETAIL OF PIONEER MINING
BLOCK OF FOUR ROOMS
PLATE 3



VERTICAL SECTION OF 26FT SEAM
SHOWING
CUTTING AND DRILLING TOP COAL
PLATE 4

amount of pick work to properly trim top-coal "lips," which is expensive. After the first bench of top coal is removed, the drilling has been done from the top-coal piles with hand-held drills.

The proposed method of top-coal extraction provides for the usual sacrifice of four feet of top coal with the "lips" to be prepared by application of a vertical and slant cut with the 10 RU machine as illustrated in Plate 4. The "lip" cuts will be made before the machine is removed from the room. It will no doubt be necessary to adequately support the "lips" after cutting by use of timber.

We propose to drill slant holes as shown with the CD-25 mounted drill on approximate four-ft centers in each direction. The method of loading top coal holes is shown in Plate 4, the proposed method being to load the back or top end of the hole with three sticks of permissible powder, stem same, then load three more sticks and stem the lower end of the hole. We believe that this can best be accomplished by use of a trough in loading the holes. There will be two detonators per hole with five holes across the room, allowing the use of a permissible ten-shot shooting battery. We feel that the placing of the powder as shown and shooting to an open end will make possible good preparation and elimination of large chunks usually found in top coal shooting. It is necessary to eliminate large chunks due to the use of belt-conveyor haulage.

The top coal will be cut, drilled, and shot down in blocks 100 ft long, furnishing approximately 2500 tons per fall for continuous loading.

Returning to Plate 1, it is important to describe the proposed sequence of top coal extraction in rooms and room slant crosscuts. The top coal in the room proper will be extracted back to within approximately 20 ft of the lower rib of the slant crosscut above, leaving a bridge of top coal to support the narrow pillar at this point. The top coal is then pulled back in the slant crosscut to the high rib of the room and another block of top coal extracted in the room proper. All top coal is removed from the block of four rooms before abandonment permitting the sealing of an individual block of four rooms if it becomes necessary. The solid pillar between each block of four rooms is left for sealing purposes.

Return to Plate 3, illustrating a continuous cycle of operation of cutting, drilling, and loading. The labor personnel required is set forth and is adequate to maintain a complete and continuous cycle of operation in Pioneer room mining and top-coal extraction. The three men indicated as timbermen or trimmers, together with the electrical and mechanical repairmen, will execute dead work, i.e. necessary timbering, extension of line curtains, trimming of "lips" and ribs, etc. Our projected or anticipated production from Pioneer room mining is the cutting, drilling, and loading of seven places or 500 tons per shift. Our anticipated production from top coal is 800 tons per shift. There are many obstacles to overcome in mining pitching seams not common to flat seam mining and the projected or anticipated production set forth above is therefore conservative with expected improved performance after experience is gained.

We foresee the principal advantages of the trackless mining system applied to our particular condition to be as follows: (1) elimination of track in panel and rooms; (2) elimination of panel development hoist; (3) elimination of panel hoist installation; (4) saving on power consumption affected by use of a 40-hp belt conveyor in lieu of 200-hp panel hoist facility; (5) mine labor saving on panel development due to more efficient operation; (6) labor saving on Pioneer room and top-coal mining due to more efficient operation.

Taking into consideration the above six major items of indicated advantage over the present system, we have projected a saving of approximately 16 cents per ton against the 480,000 tons to be removed from each panel. The capital outlay for equipment is approximately $150,000, bringing about an additional depreciation charge per ton based on a fifteen-year period, and 200 working days per year of approximately 4 cents per ton which must be deducted from the indicated saving of 16 cents per ton, leaving a net saving of 12 cents per ton expected to result from the projected system as compared to the system now in use. We are convinced that this can be accomplished.

Stock Piling— Past, Present, and Future

RICHARD J. LUND • SUPERVISOR, ENGINEERING ECONOMICS DIVISION, BATTELLE MEMORIAL INSTITUTE; MEMBER, AIME

Stock piling—and by that I mean well-organized stock piling on a substantial scale—is almost as old as the hills themselves. It was back in early Biblical times, as recounted in the Book of Genesis, that Joseph was charged by Pharaoh with the responsibility of stock piling food during the seven years of plenty to guard against the seven subsequent years of famine that were revealed by the Pharaoh's dream. And this same policy of storing up food or materials to provide for future anticipated emergencies can be traced repeatedly down through the ages. So our own efforts are anything but new.

Our modern concept of stock piling—the purchase and storage of strategic materials to assure supplies needed in time of some future war—stems from our experience in World War I. In 1920, the War Department was charged by Congress with the responsibility of procuring all War Department supplies and mobilizing industry in the event of another war. During the two decades that followed there were many authoritative studies made of our deficient mineral problem, all of which pointed out the urgent necessity of stock piling certain strategic materials as a reserve against future war. One such study, made in 1925 through the combined efforts of the Mining and Metallurgical Society and the AIME, was done at the express request of the War Department. Eight years later another more detailed authoritative report advocating the stock piling of one of the most vital of strategic minerals, manganese, was released by the AIME Industrial Preparedness Committee, and in the same year stock piling was urged in one of the many excellent papers prepared by The Mineral Inquiry, organized by the AIME. Shortly thereafter stock piling was again strongly recommended by the President's Planning Committee for Mineral Policy in the National Resources Board report of Dec. 1, 1934. Special unpublished studies were made later by the Bureau of Mines and other government organizations. Just prior to 1940 the Army and Navy Munitions Board received reports on strategic minerals from its Minerals Advisory Committee of civilian experts, many of which strongly advocated stock piling.

There was no dearth of expert advice during the '20s and '30s regarding the urgent need of stock piling for national insurance.

Yet, in spite of the fact that the War Department and Munitions Board long advocated the principle of stock piling, it was not until 1937 that official acceptance of the principle was implemented by including a small allotment of funds for this purpose in the Navy Department Appropriation bill for 1938. In all fairness in this retrospect judgment, however, we should recall that those decades were times of disarmament and military retrenchment, followed by years of severe depression when military expenditures were anything but popular.

start made in 1939

Finally, under the threat of war in Europe, a real start was made toward stock piling when Congress passed the Strategic Materials Act in June, 1939. Even though the program was started much too late, with objectives far too low for too few materials and carrying specifications too rigidly maintained at high levels, substantial progress was made during the midst of the greatest war in history in building up sizable stocks of a long list of strategic or critical materials. Stock-piling problems during this period were: (1) mushrooming in numbers of materials covered in the program, one list containing over 150 separate items; (2) during the early war years the constant upward revision, to fantastic heights, of objectives believed necessary to provide, first, a two-year emergency period, and later, a three-year period. Later came retrenchment, when greatest concern was not over having enough in the stock piles, but over having too much.

The wartime stock-piling program involved two phases: (1) a so-called "public purchase" program, covering both imports and domestic production of materials ranging from high to low quality, with stocks constantly being released to industry for essential uses; and (2) a stock-pile reserve, consisting of the residual after

This paper was presented by the Mineral Economics Division at the 1949 AIME Annual Meeting.

quantities for current use were satisfied.

stocks sacrificed for postwar prosperity

Even prior to victorious termination of World War II the philosophy of emphasizing redistribution of the stock piles so painfully accumulated during the war was translated into law as Section 22 of the Surplus Property Act of 1944. Under this act all government-owned supplies of strategic minerals were to be transferred to the permanent stock pile, *only* when declared surplus. That is, if consumers needed or wanted these materials they were given first call with only the residue to be eligible for permanent stock piling. Even considering the victory-flavored spirit of the times, it is difficult to conceive how the wartime material supply problems and troubles could be so quickly and completely forgotten by those responsible for legislating and administering stock-piling matters. Many of our fairly sizable stocks of strategic materials were literally legislated away as a sacrifice to peacetime prosperity within a period of a little over two years.

Following extensive hearings and debate, the Stockpiling Act of 1946 became law. Incorporated in this were essentially the same provisions for transfer of surplus stocks held by Government agencies to the permanent stock pile, exempting from the transfer requirements "such amount of any material as is necessary to make up any deficiency of the supply of such material for the current requirements of industry as determined by the Civilian Production Administration or its successor." Depletion of stocks of many metals or minerals continued under this provision.

postwar progress

Until mid-1948, progress in building up stocks of newly acquired materials under this Act was poor, as stated publicly by responsible officials. But it must be recognized that accumulation of stocks in a boom period when supplies of many of the raw materials involved are insufficient even to meet industrial needs poses a serious problem. The policy until last year was to avoid active competition with industry for limited supplies, giving plenty of weight to the stipulation in the Act that purchases "shall be made, so far as is practicable, from supplies of materials in excess of the current industrial demand." Apparently not too much weight was given to the progressive deterioration of world peace efforts between 1946 and 1948 in interpreting the phrase "so far as is practicable."

Table 1. Progress in Stock Piling, June 30, 1948

	Millions of dollars	Percent
Gross delivered value of stock pile objectives	3403	100.0
Material on hand:		
From Public Law 117 stock pile	86	2.5
From surplus property	424	12.5
From Public Law 520 deliveries	87	2.6
Total on hand	597	17.6
Balance of requirements not on hand	2806	82.4
Materials in sight:		
Further surplus property	29	0.9
Deliveries against present contracts	223	6.5
Total in sight	252	7.4
Balance to be purchased	2554	75.0
Unobligated funds available	490	14.4
Funds not yet available	2064	60.6

Progress in stock piling as of mid-1948 is shown in Table 1, taken from the Munitions Board's Nonconfidential Supplement to the Stockpiling Report to Congress, July 23, 1948. Only one quarter of the objective was on hand or in sight under contracts then in force, and sixty percent of this amount was from transfers or from the Public 117 stock pile. Furthermore, stocks were admittedly in an undesirable state of unbalance. With the Berlin crisis and other threats that the cold war might at any time burst into flame came a belated recognition that "the stock pile is not a marginal program nor a residual claimant for resources" but is actually "a high priority program, a vital part of the pattern for national security," as stated in the Jan. 23, 1949, Stockpile Report to Congress. Since then progress has been accelerated, and results happily are beginning to appear.

At the end of 1948 the value of materials on hand was $821 million comprising $66 million from the prewar stock pile, $584 million from surplus property transfers, and $171 million from purchases with funds appropriated since 1946. The speed-up in purchasing activity is plainly indicated by figures given in the same report showing the history of stock-pile obligations and expenditures under the Public Law 520 program. These are reproduced in Table 2.

The substantial step-up in the purchasing program during the last half of calendar 1948 is striking. In appraising the increase in obligations and expenditures, however, it must be remembered that these years cover a period of rapidly rising prices, and the figures should be discounted by this factor in judging increases in physical volume of purchases. It is also interesting to note from this report that over the entire period a total of 1834 purchases were made at an average per purchase of about $378,000. Prior to mid-1948 only four long-range contracts, three to five years, had been made at a total cost of about $25 million. In the last half of 1948, ten more long-term contracts were signed at a total of over $183 million. Average size of these long-

Table 2. Stock-pile Obligations and Expenditures Under Public Law 520, By Six-Month Periods (In Millions)

Period	Obligations Incurred Total	Obligations Incurred Monthly Average	Expenditures Total	Expenditures Monthly Average
Fiscal Year 1947				
July-December 1946	$ 34.9	$ 5.8	$ 0.0	$ 0.0
January-June 1947	31.7	5.3	9.1	1.5
Fiscal Year 1948				
July-December 1947	117.0	19.5	23.1	3.9
January-June 1948	126.7	21.1	54.6	9.1
Fiscal Year 1949				
July-December 1948	402.2	67.0	92.6	15.4

term contracts was thus almost tripled, from $6 million to about $18 million. Many other generalized data are given indicating progress. An important factor in this accelerated program was the substantial increase in funds made available by Congress.

As of the end of 1948 virtually the entire $800 million of cash and contract authority authorized under Public 520 had been spent, committed, or earmarked, leaving nothing available for new purchases. The President has asked for supplemental authorizations of $310 million for the remainder of fiscal 1949, together with $525 million for new obligational authority in 1950. Thus every effort is being made to keep the program moving at an increased tempo. He estimated that of the total stock-pile objective, now put at $3.7 billion, materials and authorizations totaling $2.3 billion will have been provided by the end of fiscal year 1950, and that deliveries by then should be about $1.6 billion. This would amount to an increase of about $1 billion in deliveries during the fiscal years 1949 and 1950.

ECA hopes still unfulfilled

With the passage of the Foreign Assistance Act of 1948 in April of last year, expectations seemed justified that imports of strategic minerals for stock piling could be materially increased. This was to be accomplished through effective administration of certain provisions of the ECA Appropriations Act which specified that at least five percent of all grants, after conversion to the currencies of the recipient countries, shall be earmarked for purchase of scarce materials for the United States. It is still too early to see any tangible results from the ECA program, and no well-documented official statements have been released concerning progress. Recent reports, however, indicate some serious obstacles are interfering with their program to expand foreign production of strategic minerals—namely, lack of sufficient funds in the right places, insufficient long-range purchasing power, and a "vague Congressional policy." Apparently the necessity of negotiating really long-term contracts, up to, say, ten years, is causing the rub, with the possibility that in less than that time the stock-pile objective will have been reached. The mineral remaining under contract would have to be sold in competition with private industry or provided for in some new type of stock pile. The question is rightly asked, "What sort of protection will be afforded to the domestic mining industry against such quantities of materials being dumped on the market?"

A possible answer to this may lie in the fact that there are minimum and maximum stock-pile objectives, as explained in various exhibits in the Lemke Committee report on stock piling. The Jan. 23, 1949, Munitions Board report states plainly that "The objectives of the program are minimum ones."

The explanation of the assumptions on which the stock-pile computations are based, as given in the Lemke report, would appear to indicate the probability that the difference between minimum and maximum objectives might be substantial. In fact, in an article entitled "Our Phantom War Stockpiles" in the December 1948, *Nation's Business*, Hippelheuser stated that whereas the *minimum* objective totaled $3.4 billion, the *maximum* objective was more than double that figure, or around $8 billion. Since the present procurement plans are based on *minimum* objectives to be attained in five years—and a past chairman of the Munitions Board testified in March, 1948, that they were then behind on that five year schedule—it is quite obvious that the greater security from *maximum* stock-pile objectives would keep the stock-piling program going for something in the neighborhood of ten years.

Hippelheuser also asserts that there has been for some time quite a heated controversy over the question whether *maximum* or *minimum* objectives should be used in stock-piling plans, with prominent mineral experts in the Bureau of Mines championing the *maximum* and the Joint Chiefs of Staff insisting on the *minimum* objectives. The difference in viewpoint, according to Hippelheuser, revolves largely around the extent to which supply lines, and hence foreign sources of materials, can be counted on in time of war, the so-called "ground rules of availability." The *maximum* stock pilers advocated reliance on only very limited life lines; the *minimum* stock pilers put their faith in being able to keep supplies coming from greatly extended life lines. In considering "ground rules" and other vital matters pertaining to defense, we must realize fully the changed status of world affairs since World War II. We must realize that we are now pretty much on our own, with serious doubt as to how much effective help we can expect from other potential allies, at least in making the elaborate preparations necessary to fight and win another war.

But even if stock piles were completed before long-term contracts negotiated by ECA were fulfilled, is there any dire threat involved to domestic mining? In the case of all items on the stock-piling list, substantial proportions of our total normal peacetime supplies must be met by imports. There is little doubt that five to ten years hence our dependence on imports will be no less, with a good chance it will be greater as depletion of our own resources continues. By then some of our present foreign supply sources may also have been mined out. Thus, present ECA action resulting in development of new or expanded foreign production may in reality be only "jumping the gun" by a few years on action that otherwise would have to be taken by industry—providing us, meanwhile, with vitally needed and adequate national insurance.

domestic program needed

Let it not be inferred that I am advocating stock piling only by importation from foreign sources. The exploration for and development of new or expanded output for stock piling from domestic sources are certainly of equal importance, with special emphasis attaching to research on means of utilizing most economically low-grade domestic deposits requiring beneficiation. The active participation of the Geological Survey and Bureau of Mines in such a program is necessary.

It would appear advisable to the writer to make use of long-term contracts to encourage production from specific marginal domestic deposits. Perhaps additional legislation aimed directly at encouraging and assisting

exploration would also be desirable. It goes without saying that any general revision in government laws and regulations—whether relating to taxes, tariffs, security regulations, or other matters—resulting in a fuller recognition of the highly specialized and speculative nature of the mineral industry will help materially.

From public statements made in 1948 by various officials concerned with the government stock-piling program, especially the recent Stock-pile Report by the Munitions Board, it is evident that growing international tensions have resulted in action leading toward an acceleration of stock piling. Steps necessary to accomplish this, already implemented at least in part, are:

(1) Adoption of a more aggressive purchasing policy, with more emphasis on the vital importance of stock piling as compared with the need of meeting industry's marginal requirements.

(2) Co-operation with industry in obtaining materials through voluntary agreements, thus helping to minimize open competition in crowded markets for limited supplies.

(3) Stimulation of foreign and domestic production by using long-term purchase contracts. In making such contracts, best advice obtainable should be sought and heeded in dealing only with dependable contractors, so that funds are not tied up in nonproductive deals.

(4) Granting by Congress of adequate funds and passage of any additional legislation to permit effective implementation of steps (1) through (3).

(5) Freezing of all RFC stocks of materials useful in an emergency, for which any provision for stock piling has been made, and transfering these to the permanent government stock pile. It is fantastic to think of taking something out of one pocket at the same time that costly efforts are being made to put the same thing in the other pocket. However, this now is tantamount to locking the barn door after the horse has been stolen.

(6) Careful review of material specifications, and the adoption of a greater degree of flexibility in their application. It is encouraging to note in the last Munitions Board report that progress is being made in this direction.

With reference to item (2) above—namely, co-operative purchasing arrangements with industry—action by the Munitions Board in openly announcing their purchase objectives for a part or all of fiscal year 1949 is highly commended. This was done, with the co-operative advice of the National Association of Manufacturers and other trade association groups, "to clear up confusion and speculation as to what the government needs, and to help industry aid more effectively."[5]

There remains for consideration one more point. According to a Bureau of Mines document published in the Lemke Stock-pile Report,

> "Emergency requirements were computed on the basis of the peak year established during the recent war, including military, lend-lease, and civilian use. For some commodities the requirements estimate was adjusted to reflect changes in future military consumption resulting from technologic changes already evident and to compensate for the inferior grades of material that had to be used during the last war."

The Jan. 23, 1949, Munitions Board report, incidentally, reveals that such requirements are also based on an assumed five-year period of war.

It is vitally important that constant appraisal and reappraisal be made of future military requirements, viewed realistically, and that significant changes be promptly reflected in modified stock-pile objectives. It is equally important to maintain close surveillance over the assured supply picture, since this has an equal bearing on the stock-piling program.

In this connection one more point bears emphasis. The shortage of electric power—both now and in at least the nearby future—is acute. There is little doubt that power would be a severe bottleneck in any nearby future emergency. Electricity, as such, cannot be stored in substantial quantities but it can be effectively stored in the form of stock-piled aluminum, magnesium, or ferroalloys—materials that require large amounts of power for conversion from ore to metal. For this reason I add the plea that serious study and consideration be given to converting substantial portions of the ores of these metals now being stockpiled to the metals themselves or to their ferroalloy forms. A recent revision in the definition of "strategic and critical materials" now makes this possible. Furthermore, such a procedure would provide additional insurance against the effects of damage through enemy action to a relatively few vitally important conversion plants in time of war. While the cost involved in this conversion is admittedly high, it is nothing like the differentials of 22 times for manganese or of twenty times for vanadium as mentioned in the last Stock-pile Report. In fact, according to current quotations the manganese ore conversion amounts to a factor of only about three, and vanadium ore of about six.

In closing, I can only reiterate the urgent need—recognized, I am sure, by all of us and by every intelligent American—for continued accelerated effort in attaining realistic stock-pile objectives at the earliest practicable time. Money spent on this program is a wise investment in national security, and even though some measure of harm or inconvenience may be suffered by segments of the mineral industry in the program speed-up, such sacrifices should and undoubtedly will be made in the national interest.

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ALL INSTITUTE SECTION

APPEARING IN:
MINING ENGINEERING
• JOURNAL OF METALS
• JOURNAL OF PETROLEUM TECHNOLOGY

More Students in Mineral Engineering Courses

By WILLIAM B. PLANK AND EDWARD MARTINEZ • HEAD DEPARTMENT OF MINING AND METALLURGICAL ENGINEERING AND ASSISTANT PROFESSOR IN MINING, RESPECTIVELY, LAFAYETTE COLLEGE.

There were 15,028 students enrolled in mineral engineering courses on Dec. 1, 1948, in the United States and Canadian colleges, as revealed by the ninth biennial questionnaire study made by the writers under the auspices of the Mineral Industry Education Division of the AIME. Sixty-four colleges in the United States and eight in Canada are covered by the study.

Every mineral engineering school in both countries is represented in the study and for that reason the data are comparable with former studies. As formerly, the general enrollment figures are included, taken this year from the February, 1949, issue of the *Journal of Engineering Education*.

The combined enrollment of 15,028 mineral engineers in both countries is 16.6 percent larger than the enrollment of 12,892 in 1946-1947. There was an increase of 20 percent in the United States figures, but a decrease of 14 percent in the Canadian figures during this period. These comparisons are shown in Tables 1 and 2.

The current total enrollment of 249,913 engineers of all kinds in the United States and Canadian colleges, shown in the data supplied by the ASEE in Table 3, is 12 percent greater than the 222,557 enrollment in 1946-47. The mineral engineering group of both countries showed a growth of 16.6 percent during the same period. As compared with 1947-48, however, the total engineering enrollment decreased about 1 percent, whereas the mineral engineering enrollment increased 22 percent to an all-time peak, thus maintaining a steady growth that was begun right after the war.

The number of first degrees in engineering conferred in 1946-47, as shown in Table 4, was 19,272 and in 1947-48, 30,018. It has been estimated that this year at least 40,000 engineers will be graduated and probably the same number will graduate next year. This year 2881 mineral engineers in both countries will receive their first degrees and the number will probably be larger next year. Thus it is seen that the deficit in the supply of engineers caused by the war is being met rapidly.

An indication of the possible trend in engineering enrollment can be seen in the decrease in the size of the freshman classes over these two years as shown in Table 3. In 1946-47, the total number of freshman engineers of all kinds in the United States and Canada was 84,652, whereas this year the group numbers 49,017. This is a decrease of 42 percent. The decrease in the number of mineral engineering freshmen during the same period, however, was from 4933 to 3090, or 37 percent.

Eventually, the decline in veteran enrollment and the lower birth rate during the war, of which these figures are an indication, will materially reduce the size of the engineering student groups of all kinds. S. C. Hollister, of Cornell University, estimates that the number of entering freshman engineers is undergoing a normal readjustment which by 1952-53 will

TABLE 1. MINERAL ENGINEERING STUDENT ENROLLMENT BY COURSES—1948-49, UNITED STATES, 64 SCHOOLS

Course	Freshman	Sophomore	Junior	Senior	Total Undergraduates	Graduate Students	Grand Total	Candidates for B.S. Degree
Mining	481	844	517	466	1,908	82	2,080	428
Metallurgy	648	626	1,008	815	3,308	552	3,860	799
Petrol. and Nat. Gas.	1,024	1,076	1,161	722	4,083	51	4,084	788
Ceramic	181	294	515	231	1,011	164	1,175	227
Geology	853	611	466	401	2,283	117	2,400	370
Fuels	0	0	0	0	31	16	47	0
Specials	.....	.....	.....	.....	83	.....	83	.....
Totals	2,884	3,330	3,686	2,663	12,667	1,102	13,769	2,596

Revision of a paper read before Mineral Industry Education Division, AIME, San Francisco, Calif., Feb. 14, 1949.

TABLE 2. MINERAL ENGINEERING STUDENT ENROLLMENT BY COURSES—1948-49, CANADA, 8 SCHOOLS

Course	Freshman	Sophomore	Junior	Senior	Total Undergraduates	Graduate Students	Grand Total	Candidate B.S. Degree
Mining	99	112	183	178	552	0	552	172
Metallurgy	43	66	99	77	287	11	298	77
Petrol. and Nat. Gas	13	23	17	0	53	0	53	0
Ceramics	6	13	6	11	37	2	39	11
Geology	36	65	76	83	260	57	317	83
Totals	196	281	383	349	1,189	70	1,259	343

TABLE 3. ENGINEERING ENROLLMENT BY CLASSES AND COURSES—1948-49, 151 UNITED STATES AND 6 CANADIAN SCHOOLS

Course	Freshman	Sophomore	Junior	Senior	5th Year and Others	Total Undergraduates	Graduate Students	Grand Total
Mining	427	576	637	827	6	2,173	83	2,256
Metallurgy	483	819	1,056	871	127	3,368	687	4,055
Petrol. and Nat. Gas.	921	1,186	1,309	916	105	4,437	96	4,533
Ceramics	161	264	319	247	95	1,086	120	1,206
Total Mineral Engineers[1] U.S. and Canada	2,002	2,845	3,323	2,561	333	11,064	986	12,050
Chemical	3,722	5,095	5,965	4,784	1,092	21,248	2,371	23,619
Civil	5,522	8,139	9,483	7,113	1,541	31,798	2,063	33,861
Mechanical	7,725	12,446	15,380	12,093	3,453	51,107	2,535	53,642
Electrical	7,416	11,830	15,725	11,449	3,496	49,907	3,867	53,774
Other Engineers	19,060	12,855	11,921	9,280	6,912	62,048	3,801	65,849
In Extension Centers	3,550	1,999	730	814	405	7,118	.....	7,118
Total Engineers U. S. and Canada	49,017	55,079	62,527	47,785	19,882	234,290[2]	15,623	249,913
Veterans, U. S. and Canada	18,282	26,011	42,959	33,909	12,183	132,394	5,880	138,274

[1] Geological and Fuel Engineers included in "Other Engineers".
[2] Includes 1210 women.

TABLE 5. ENGINEERING STUDENT ENROLLMENT BY COURSES—UNDERGRADUATES AND GRADUATES—1940-41 TO 1948-49

	(1) 1940-41	(1) 1944-45	(1) 1946-47	(1) 1947-48	(2) 1948-49	(3) 1948-49
Mining	1,978	224	2,302	2,489	2,256	2,632
Metallurgy	2,480	472	3,178	4,121	4,055	4,156
Petrol. and Nat. Gas	2,877	376	3,048	4,065	4,533	4,147
Ceramics	843	190	940	1,108	1,206	1,214
Geology	792	?	?	?	?	2,777
Fuels	44	?	?	?	?	47
Specials	149	?	?	?	?	53
Total Mineral Engineers: United States	9,133	.....	.....	.....	.....	13,789
Canada	.....	.....	.....	.....	.....	1,259
U. S. and Canada	.....	1,282	9,469	12,353	12,050	15,029
Chemical	.....	5,082	23,600	25,972	23,619	.....
Civil	.....	3,537	30,079	31,949	33,861	.....
Mechanical	.....	6,965	44,403	50,551	53,642	.....
Electrical	.....	5,934	38,983	50,406	53,774	.....
Others	.....	10,652	66,475	69,117	72,909	.....
Total Engineers	114,116	41,632	222,557*	282,250	249,913	.....

[1] Mining & Metallurgy, April 1946, U. S. and Canada.
[2] Journal of Engineering Education, Feb. 1949, U. S. and Canada.
[3] Study of Mineral Education Division, AIME, Dec. 1, 1948.
* Includes 9566 Canadian undergraduate students, unclassified.

bring the number of graduations down to the prewar gradient of steady upward growth.[1]

The percentage of veterans studying engineering in both countries dropped from 57 percent of the total last year to 55 percent this year. This year 31 percent of the freshman engineers are veterans compared to 42 percent last year.

Among the mineral engineers in the United States, the petroleum and natural gas engineers maintain their position as the largest group, with

TABLE 4. FIRST DEGREES CONFERRED, ALL ENGINEERS, UNITED STATES AND CANADA

1942-43	14,714
1943-44	11,185
1944-45	4,724
1945-46	8,219
1946-47	19,272
1947-48	30,018

4094 students. They are followed closely, however, by the metallurgists with 3860 students. Next in rank are the geological engineers and mining geologists, listed under the title "Geology" in the tables, with 2460 students. The mining engineers rank fourth, numbering 2080 students, and the ceramic engineers are fifth in rank, with 1052 students.

Of significance to employers of mineral engineers in the United States is the fact that there are available for positions in the mineral industries only 2538 bachelors and about 500 graduates from this year's class. As usual, however, not all of these men will be available for jobs since a number will go into teaching, graduate schools, or other industries.

Employment opportunities for this year's mineral engineering graduates generally have been good but with a slower demand for men in the industries affected by the present slowing up of production. Demand for mining engineers in the east is reported to be better than elsewhere especially by the iron ore and the coal industries.

The employment situation for other engineers who graduated this year has not been so favorable. Deans of engineering schools say that all graduates will be placed but not as rapidly as in recent years. It has been estimated that out of the 40,000 engineers who graduated this year only 24,000 will be absorbed by industry, leaving a surplus of 16,000. It is believed that of the many factors that may cause a sharp upward trend in engineering employment of all kinds are the wider use of technology, the use of engineers in other professions, and above all, the increasing number of engineering problems involved in winning from the earth, the sea, and the air, the raw mineral materials upon which our highly industrialized civilization depends.

[1] Postwar Engineering Enrollment, Journal of Engineering Education, March, 1949.

TABLE 6. DISTRIBUTION OF GRADUATE STUDENTS IN MINERAL ENGINEERING—UNITED STATES, 1933-34 TO 1948-49

	1933-1934	1934-1935	1936-1937	1938-1939	1940-1941	1942-1943	1946-1947	1948-1949
Mining	40	55	49	44	44	20	70	82
Metallurgy	116	93	131	172	215	137	470	552
Petrol and Nat. Gas	18	29	55	43	75	27	187	91
Ceramics	19	36	48	64	80	19	138	164
Geology	48	62	34	88	127	9	127	197
Fuels	4	10	10	6	9	10	16	16
Totals	243	285	330	415	520	222	1,008	1,102

Data from Mineral Industry Education Division, AIME, studies.

Directory of Mineral Engineering Schools in the United States and Canada

**Accredited by ECPD, October, 1948*

The name and address of the school are given, followed by the length of the regular undergraduate curriculum, the degree granted, types of courses given, and the name of the man in charge. This covers only work for the first or Bachelor degree in each case. School catalogs should be consulted for details.

The following abbreviations are used:

A. Arts
B. Bachelor
Cer. Ceramics or Ceramic
E. Engineer or Engineering
Eng. Engineering or Engineer
Eco. Economic
Geol. Geologist, Geological, or Geology
M. Mines or Mining
Met. Metallurgy or Metallurgical
Min. Mineral
Mng. Mining
M.S. Master of Science
Pet. Petroleum
Ref. Refining
S. Science
Tech. Technology or Technical

UNITED STATES

University of Alabama, University, Ala. 4-yr. B.S. in: (1) *Mng. Eng., (2) Met. Eng., (3) Met., (4) Cer., (5) Pet. Tech. James R. Cudworth, dean; J. W. Stewart, head, School of Mines; T. N. McVay, head, Dept. of Ceramics.

University of Alaska, College, Alaska. 4-yr. or 5-yr. B.S. major in: (1) *Mng. Eng., (2) *Mng. Eng., Geol. option, (3) *Mng. Eng., Met. option. Richard C. Ragle, head, Dept. of Geology; Earl H. Beistline, head, Dept. of Min. & Met.

University of Arizona, Tucson, Ariz. 4-yr. B.S. in: (1) *Mng. Eng., (2) Met. Eng., (3) Mng. Geol. (4) Geol. T. G. Chapman, dean, College of Mines.

Polytechnic Institute of Brooklyn, Brooklyn 2, N. Y. 4-yr. B.Met.E. Otto H. Henry, professor of Metallurgical Engineering.

University of California, Berkeley, Calif. 4-yr. degrees in: (1) *Mng. Eng., (2) *Met., (3) *Pet. Eng., (4) Mineral Exploration. Lester C. Uren, chairman, Division of Mineral Technology.

Carnegie Institute of Technology, Pittsburgh, Pa. 4-yr. B.S. in *Met. Eng. Robert F. Mehl, head, Dept. of Metallurgical Engineering.

Case Institute of Technology, Cleveland, Ohio. 4-yr. B.S. in *Met. Eng. K. H. Donaldson, head, Dept. of Metallurgical Engineering.

University of Cincinnati, Cincinnati, Ohio. 5-yr. *Met. Eng. Cooperative. Dr. Roy O. McDuffie, professor of Metallurgy.

Colorado School of Mines, Golden, Colo. 4-yr. (1) *E.M., (2) *Geol. Eng., (3) *Pet. Eng., (4) *Met. Eng., (5) Pet. Ref. Eng. Ben H. Parker, president.

Columbia University, New York, N. Y. 4 or 5-yr. B.S. in (1) *Mng. Eng., (2) *Met. Eng., (3) Mineral Eng. Philip B. Bucky, prof. of Mining and Exec. Officer of Dept. of Mng., Met. and Min. Eng.

Cornell University, Ithaca, N. Y. 5-yr. B.Met.E. Fred H. Rhodes, director of the School of Chemical & Met. Eng.; S. C. Hollister, dean of the College of Engr.

Drexel Institute of Technology, Philadelphia 4, Pa. 5-yr. (co-operative) B.S. in Met. Eng. A. W. Grosvenor, professor of metallurgy.

Fenn College, Cleveland, Ohio. 4½-yr. cooperative in B. of *Met. Eng. Dr. G. U. Greene, head, Dept. of Metallurgical and Chemical Engineering.

Georgia Institute of Technology, Atlanta, Ga. 4-yr. B. of *Cer. Eng. Lane Mitchell, director, School of Ceramic Engineering.

Harvard University, Cambridge, Mass. Undergraduate instruction in engineering sciences and applied physics, or geological sciences, preparatory to professional study in Graduate School of Engineering in (1) *Physical Met., or in Division of Geological Sciences of the Graduate School of Arts and Sciences in (2) Min. Geol., (3) Pet. Geol., (4) Geophysics; or in the Dept. of Engineering Sciences and Applied Physics of the Graduate School of Arts and Sciences. Marland P. Billings, chairman, Division of Geological Sciences.

University of Idaho, Moscow, Idaho. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) Geol., (4) *Geol. Eng. A. W. Fahrenwald, dean, School of Mines.

Illinois Institute of Technology, Chicago 16, Ill. 4-yr. B.S. in Met. Eng. Otto Zmeskal, director, Dept. of Metallurgical Engineering.

University of Illinois, Urbana, Ill. 4-yr. B.S. in: (1) *Cer. Eng., (2) *Met. Eng., (3) *Mng. Eng. H. L. Walker, head, Dept. of Mining and Metallurgical Engineering; A. I. Andrews, head, Dept. of Ceramic Engineering; George W. White, head of Dept. of Geology.

Iowa State College, Ames, Iowa. 4-yr. B.S. in (1) *Cer. Eng., (2) Mng. Eng. G. L. Bridger, head, Dept. of Chemical and Mining Engineering; C. M. Dodd, head, Dept. of Ceramic Engineering.

University of Kansas, Lawrence, Kans. 4-yr. B.S. in: (1) Geol. Eng., (2) Met. Eng., (3) *Min. Eng., (4) Pet. Eng. R. M. Dreyer, Chm., Dept. of Geol. & Geological Eng.; K. E. Rose, Chm., Dept. of Mng. & Met. Eng.; and C. F. Weinaug, Chm., Dept. of Pet. Eng.

University of Kentucky, Lexington, Ky. 4-yr. B.S. in: (1) *Met. Eng., (2) Mng. Eng. C. S. Crouse, head, Dept. of Mining and Metallurgical Engineering.

Lafayette College, Easton, Pa. 4-yr. B.S. in: (1) *Met. Eng., (2) *Mng. Eng. W. B. Plank, head, Dept. of Mining and Metallurgical Engineering.

Lehigh University, Bethlehem, Pa. 4-yr. B.S. in: (1) *Mng. Eng., (2) Mng. Eng., Eng. Geophysics option, (3) *Met. Eng. A. C. Callen, head, Dept. of Mining Engineering: Gilbert E. Doan, head, Dept. of Metallurgical Engineering.

Louisiana Polytechnic Institute, Ruston, La. 4-yr. curriculum in Pet. Eng. Roy T. Sessums, dean, School of Engineering.

Louisiana State University, Baton Rouge, La. 4-yr. B.S. in: (1) *Pet. Eng., (2) Pet. Geol. Henry V. Howe, director, School of Geology.

Massachusetts Institute of Technology, Cambridge, Mass. 4-yr. S.B. in *Metallurgy. John Chipman, head, Dept. of Metallurgy.

Michigan College of Mining and Technology, Houghton, Mich. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) Geol. Eng. Grover C. Dillman, president.

Michigan State College, East Lansing, Mich. 4-yr. B.S. in Met. Eng. L. G. Miller, dean of Engr.; R. L. Sweet, Chm., Met. Eng.

University of Michigan, Ann Arbor, Mich. 4-yr. B.S.E. *(Met.). George G. Brown, chairman, Dept. of Chemical and Metallurgical Engineering.

University of Minnesota, Minneapolis, Minn. 5-yr. (1) *B. Mng. Eng., (2) B. Geol. Eng., (3) *B. Pet. Eng., (4) *B. Met. Eng. T. L. Joseph, dean, School of Mines and Metallurgy.

Missouri School of Mines and Metallurgy, Rolla, Mo. 4-yr. B.S. in: (1) *Mng. Eng. with options in (a) Mng. Geol., (b) *Pet. Production; (2) *Met., (3) *Cer. Eng., (4) B.S., with major in Geol., Cer., or Met. B.S. in Chemical Eng. for those specializing in Pet. Refining. Curtis L. Wilson, dean.

Montana School of Mines, Butte, Mont. 4-yr. B. S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) *Geol. Eng., (4) Pet. Eng. Francis A. Thomson, president.

University of Nevada, Reno, Nev. 4-yr. B.S. in: (1) *Mng. Eng. (2) Met. Eng. (3) Geol. Eng. Jay A. Carpenter, director, Mackay School of Mines.

New Mexico School of Mines, Socorro, New Mex., 4-yr. B.S. in: (1) *Mng. Eng., (2) Met. Eng., (3) *Pet. Eng., (4) *Geol. Eng., (5) Geophysics, (6) Chemistry. 5-yr. B.E. in: (1) Geology. E. J. Workman, president.

New York State College of Ceramics, Alfred, N. Y. 4-yr. B.S. in: (1) *General Cer. Tech. and Eng., (2) Glass Tech. J. F. McMahon, Acting Dean.

North Carolina State College of the University of North Carolina, Raleigh, N. C. 4-yr. B. of: (1) *Cer. Eng., (2) Geol. Eng. W. W. Kriegal, professor of ceramic engineering; J. L. Stuckey, professor of geological engineering.

University of North Dakota, Grand Forks, N. Dak. 4-yr. B.S. in *Mng. Eng. L. C. Harrington, Dean, College of Engineering, director, Division of Mines and Mining Experiments.

University of Notre Dame, Notre Dame, Ind. 4-yr. B.S. in *Met. Edward G. Mahin, head, Dept of Metallurgy.

The Ohio State University, Columbus, Ohio, 5-yr. (1) *B. Eng. in Mng. with option in Pet. Eng., (2) B. *Cer. Eng., option in Glass Technology, (3) B. *Met. Eng. H. F. Nold, professor of mine engineering; J. L. Carruthers, head, Dept. of Ceramic Engineering; Mars G. Fontana, head, Dept. of Metallurgy; E. V. O'Rourke, professor of petroleum engineering.

University of Oklahoma, Norman, Okla. 4-yr. B. S. in: (1) Geol. Eng., (2) *Pet. Eng., (3) Natural Gas Eng. W. H. Carson, dean, College of Engineering.

Oregon State College, Corvallis, Oreg. 4-yr. B.S. in Mng. Eng., with options in (1) Mng. Eng., (2) Met. Eng. W. E. Caldwell, professor of mining engineering.

Pennsylvania State College, State College, Pa. 4-yr. B.S. in: (1) Geol. & Mineralogy, (2) Geophysics & Geochemistry, (3) Meteorology, (4) Geography, (5) Mineral Eco., (6) *Mng. Eng., (7) Mineral Preparation Eng., (8) *Pet. & Nat. Gas Eng., (9) Fuel Tech., (10) *Met., (11) *Ceramics, Edward Steidle, dean, School of Mineral Industries.

University of Pennsylvania, Philadelphia 4, Pa. 4-yr. B.S. in Met. Eng. R. M. Brick, director, Dept. of Metallurgy.

University of Pittsburgh, Pittsburgh, Pa. 4-yr. B.S. in: (1) *Mng., (2) *Met. Eng., (3) *Pet. Eng., (4) Geol. Eng. E. A. Holbrook, dean, Schools of Engineering and Mines.

Princeton University, Princeton, N. J. 4-yr. Geol. Eng., with degree B.S. in Eng. Kenneth H. Condit, dean, School of Engineering; W. T. Thom, Jr., chairman, Dept. of Geological Engineering.

Purdue University, West Lafayette, Ind. 4-yr. B.S. in *Met. Eng. G. M. Enos, chairman, Division of Metallurgical Engineering, School of Chemical and Metallurgical Engineering.

Rensselaer Polytechnic Institute, Troy, N. Y. 4-yr. B. *Met. Eng. W. F. Hess, head, Dept. of Metallurgical Engineering.

Rutgers University, New Brunswick, N. J. 4-yr. B.S. in Cer. John H. Koenig, director, Dept. of Ceramics.

South Dakota School of Mines and Technology, Rapid City, S. Dak. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) Geol. Eng. Warren E. Wilson, president.

University of Southern California, Los Angeles, Calif. 4-yr. B.S. in: *Pet. Eng. C. R. Dodson, head, Pet. Eng. Dept., Thomas Clements, head, Dept. of Geology.

Stanford University, Stanford, Calif. 4-yr. B.S. in *Min. Sciences. A. I. Levorsen, dean, School of Mineral Sciences.

Agricultural and Mechanical College of Texas, College Station, Texas. 4-yr. B.S. in *Pet. Eng. Also 5-yr. (1)* B. Pet. Eng., with options in: (a) Production, (b) Geol., (2) *B.S. in Pet. Eng. and Mech. Eng., (3) *B.S. in Pet. Eng. and Geol. Eng. Harold Vance, head, Dept. of Petroleum Engineering.

University of Texas, Austin, Texas. 4-yr. B.S. in *Pet. Eng. H. H. Power, chairman, Dept. of Petroleum Engineering.

University of Texas, College of Mines and Metallurgy, El Paso, Texas. 4-yr. B. S. in Mng. Eng. with options in: (1) *Mng. Eng., (2) Met., (3) *Mng. Geol. Wilson H. Elkins, president; E. M. Thomas, dean of Engineering.

University of Tulsa, Tulsa, Okla. 4-yr. B.S. in: (1) *Pet. Production Eng., (2) *Pet. Refining, (3) Pet. Geol., (4) Geophysical Eng. R. L. Langenheim, dean, College of Petroleum Sciences & Engineering.

University of Utah, Salt Lake City, Utah. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) Geol. Eng. C. J. Christensen, dean, School of Mineral Industries.

Virginia Polytechnic Institute, Blacksburg, Va. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) Geol., (4) *Cer. Eng. E. B. Norris, dean of Engineering.

State College of Washington, Pullman, Wash. 4-yr. B.S. in: (1) Geol., (2) *Mng., (3) *Met. Donald L. Masson, acting director, School of Mines.

University of Washington, Seattle, Wash. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng., (3) *Cer. Eng. Drury A. Pifer, director, School of Mineral Engineering.

Washington University, St. Louis, Mo. 4-yr. B.S. in *Geol. Eng. Lawrence E. Stout, dean, School of Engineering.

West Virginia University, Morgantown, W. Va. 4-yr. B.S. in Mng. Eng., with options in: (1) *Mng. Eng., (2) Oil and Gas Eng. G. R. Spindler, director, School of Mines.

West Virginia Institute of Technology, Montgomery, W. Va. 5-yr. B.S. in Mng. on co-operative system in coal. Edward L. Holt, head, Dept. of Mng. and Geology.

University of Wisconsin, Madison, Wis. 4-yr. B.S. in: (1) *Mng. Eng., (2) *Met. Eng. Geo. J. Barker, chairman, Dept. of Mining and Metallurgy.

Yale University, New Haven, Conn. 4-yr. B. Eng. in *Met. C. H. Mathewson, chairman, Dept. of Metallurgy.

CANADA

University of Alberta, Edmonton, Alta. 4-yr. B.S. in Mng. Eng., with options in: (1) Metal Mng., (2) Coal Mng. K. A. Clark, professor of metallurgy; E. O. Lilge, professor of ore dressing; T. H. Patching, asst. prof. of Mng. Eng.

University of British Columbia, Vancouver, B. C. 5-yr. B. of Applied S. in: (1) Mng. Eng., (2) Met. Eng., (3) Geol. Eng. F. A. Forward, head, Dept. of Mining and Metallurgy; M. Y. Williams, head, Dept. of Geology and Geography.

Laval University, Quebec. 4-yr. B.S. in: (1) Geol., (2) Mng. Eng. (3) Met. G. Letendre, director, Dept. of Mining and Metallurgy.

McGill University, Montreal, P. Q. 4-yr. B. Eng. in: (1) Mng. Eng., (2) Met. Eng. J. J. O'Neill, dean of Engineering.

Ecole Polytechnique (Montreal University), Montreal, P. Q. 5-yr. B. Applied S. in: (1) Mng. Eng. and Geol., (2) Met. Eng. P. Mauffette, head, Dept. of Mining and Geology. L. Bourgoin, head, Dept. of Metallurgy.

Nova Scotia Technical College, Halifax, N. S. 5-yr. B. Eng. in Mng. Eng. A. E. Flynn, professor of mining engineering.

Queen's University, Kingston, Ont. 4-yr. B.S. in: (1) Mng. Eng., (2) Met. Eng., (3) Economic Geol. A. V. Corlett, head, Dept. of Mining Engineering; T. V. Lord, head, Dept. of Metallurgical Engineering; E. L. Bruce, head, Dept. of Geology.

University of Toronto, Toronto, Ont. 4-yr. B. Applied S.: (1) Mng. Eng., (2) Met. Eng., (3) Mng. Geol. R. E. Barrett, professor of mining engineering; L. M. Pidgeon, professor of metallurgical engineering; E. S. Moore, head, Dept. of Geological Sciences.

Board Acts on Publications at June Meeting

Meeting on June 15, the AIME Board of Directors, with President Young presiding, accepted with much satisfaction the report of the tellers in the recent dues referendum, as published in the July journals. Amendment of the Bylaws effecting the increase will be voted at a Board meeting on July 27. Also, the report of the Nominating Committee, nominating Donald H. McLaughlin as President in 1950, was accepted. The names of the nominees were published last month. The Board named Gail F. Moulton as the ninth candidate for Director, inasmuch as otherwise the Petroleum Branch would have no representative in the New York metropolitan area. Candidates for Divisional offices, as printed in the July journals, were announced except that R. W. French is named as a Vice-Chairman of the Petroleum Division instead of a member of the Executive Committee.

Prices were set, on the basis of orders so far received and expected future sales, for the three volumes of Transactions to be issued early in 1950, consisting of the material in the Transactions sections (Section 3) of the journals in 1949. An opportunity was provided on the 1949 dues bill and selection slip for ordering these volumes, and further copies will be available as long as the supply lasts. The price will be $3.50 to members including Student Associates for a first copy of any or all of the three volumes; additional copies will be billed at the nonmember price of $7 each. These volumes will be known as "Transactions, AIME, 1949," thus duplicating in date the volumes authorized for publication this year, but the volume numbers will be different and it is thought that in future it will be desirable to have the volumes carry the date of publication in the journal.

The three journals will be available next year on the same basis as in 1949: one free subscription to one journal as part consideration of dues, with one subscription to either or both of the other journals at $4 each. Additional subscriptions to one journal will be billed at the nonmember subscription price of $8 per year, or $9 to foreign countries outside of the Americas.

Authorization was voted to follow the recommendation of the Seeley W. Mudd Memorial Fund Committee to send to all new Junior Members who have not received any of the Mudd books, and to future new Junior Members, one copy each of "Seventy-five Years of Progress in the Mineral Industry" (75th AIME Anniversary volume), and "A Professional Guide for Junior Engineers," by William E. Wickenden, recently published by the Engineers Council for Professional Development. Also, any one of the following volumes will be supplied free on request: "Modern Uses of Nonferrous Metals," "75 Years of Progress in Iron and Steel," "A Brief History of the Science of Metals," and "Coal Through the Ages."

Other recommendations of the Mudd Committee accepted included: (1) Building up the "AIME Series" to twelve or fifteen volumes, including one of specific interest to each of the professional Divisions of the Institute. (2) Promoting the sale of the volumes through increased publicity. (3) Printing of 3000 copies of the revised edition of "Coal Preparation," to be sold to members and Student Associates at $4 for a first copy, $8 to nonmembers and for additional copies to members, and $6 in lots of ten or more. (4) R. Dawson Hall to be editor of the previously authorized volume on health and safety in mining.

At the Annual Meeting in San Francisco, the resignation of the H. W. Johnson Committee, appointed more than a year earlier and whose report had been made at the Annual Meeting in 1948, was received. The Board had been loath to accept it, and the matter had been discussed at successive Board meetings this spring. The Committee had been especially invited to appear before the Board at its June meeting to discuss the progress so far made in adopting the Committee's recommendations. Mr. Johnson and Messrs. Leighton and Ball, however, felt that the resignation should stand, so it was accepted, with full appreciation of the excellent report that the Committee had made, on which the recent and present reorganization of the Institute structure, practice, publication policies, and personnel has been based.

In the effort to reduce costs of Institute operation, and to improve its service, Mr. Head gave a progress report on plans to survey the office procedure of the AIME and the other Founder Societies; Mr. Beall gave data on possible savings that might result from splitting the personals into Section 1 of the appropriate journals; and Mr. Appleton reported on an investigation of the possibility of reducing printing costs. It was voted to continue publication of the personals in the All-Institute section for the rest of 1949.

Selection of Francis H. Brownell as Rand Medalist for 1950 was approved. Gail F. Moulton was appointed to serve out Mr. Roosevelt's term as representative on United Engineering Trustees. AIME representatives at the First Pan-American Engineering Congress in Rio de Janeiro, July 15-24, were named as follows: Fred T. Agthe, W. H. Carson, John H. Evans, Sherwin F. Kelly, and Henrique Capper Alves de Souza. Additional representatives at the Empire Mining and Metallurgical Congress in Great Britain, July 9-23, were named as follows: P. J. Cutting and W. E. Wrather. William J. Coulter and Lloyd E. Elkins were named as representatives at the 75th anniversary celebration of the Colorado School of Mines, Sept. 29 to Oct. 1. New Affiliated Student Societies were recognized at the University of Pennsylvania—the Metallurgical Engineering Society, and at Penn State—the Petroleum Engineering Society. Progress was reported in organizing the Mining, Geology, and Geophysics Division of the AIME, with the assistance of E. R. Borcherdt.

Directors present included: Messrs. Young, Bowles, Daveler, Head, Kinzel, Kraft, Millikan, Pehrson, Rhines, Schumacher, Sullivan, Suman, Swainson, Swift, Wrather, and Fletcher, with H. P. Greenwald as alternate for N. G. Alford. The meeting, which adjourned at 4:10 p.m., was preceded by an informal luncheon at 12:30.

Mid-Year Meeting . . .

Committeemen for the Mid-Year Meeting pose, with Scott, Faust, Mueller, Johnson, Nakervis, and MacQuigg in the front row, and Westerman, Anderson, Bowers, Pfoor, Melvin, Snavely, and Deubner backing them up.

They wouldn't fit in one picture, so seated are: O'Rourke, Bowen, Reilly, Jones, Spretnak, Salsich; and standing: Hatch, Greenidge, MacIntosh, Cunningham, Woodward, Pilcher, Reed, Alkire, and Advent.

More Important Sessions

Reports are still coming in from Division Chairmen and committee heads regarding plans for the Mid-Year Meeting to be held in Columbus, Ohio, from September 25-28. A detailed program will be printed in the September issues of the Institute Journals.

industrial pollution session

Recognizing the increasing importance of air pollution problems, especially as related to the mineral industries, the Coal Division's Program Committee has scheduled an afternoon of discussion of the topic for Wednesday, Sept. 28. Elmer Kaiser, Committee Chairman, has announced that four nationally known authorities will address the symposium.

L. C. McCabe, formerly with Los Angeles County, and now with the Bureau of Mines in Washington, will discuss "Air Pollution by Industrial Gases and Liquids," and T. C. Wurts will present a paper on the "Air Polluting Solids of Allegheny County, Pa." Allegheny Co., in May, adopted new air pollution regulations, evolved by industry itself, to help cope with the local problem. Discussion will be conducted by H. F. Hebley, of the Pittsburgh Consolidated Coal Co., well-known as a pioneer in anti-pollution work, and W. C. L. Hemeon, noted authority with the Industrial Hygiene Foundation.

Members are urged to attend this very timely and important session.

The Women's Committee affairs are handled by Mesdames MacIntosh, Kaiser, Burnhagen, Anderson; back row, Mesdames Mueller, Elsea, Pilcher, Scott, and Manning.

students urged to attend forum

Another lively student forum will be held during the Mid-Year Meeting. Armed with experience gleaned from forums at El Paso and San Francisco, the young men will face prospective employers in what promises to be a highly informative session.

Industrial leaders are prepared to answer a lot of questions from this year's crop of graduates and students, especially in view of the dearth of jobs available these days. C. E. Bales, vice-president of the Ironton Fire Brick Co.; M. D. Cooper, vocational training manager, National Coal Assn.; and James Hyslop, executive vice-president, Hanna Coal Co., will be among those attending. Person-to-person contact with employers, plus a closeup view of AIME activities, will prove valuable to students. A special student rate for attending the meeting is expected.

Students are urged to seek out plane, train, bus, and hitch-hiking schedules before September. *Professors will please contact their students now, and urge their attendance.*

. . . Columbus, Ohio, September 25 to 28

The Drift of Things

. . . as followed by EDWARD H. ROBIE

More Government Control Threatened

Many officials of companies engaged in the extraction of our country's mineral wealth are alarmed at the implications of an extension of authority assumed by the Federal Power Commission toward the producing, gathering, and marketing functions of the natural gas industry. For years the majority opinion of the Commission has been that it had no powers under the Natural Gas Act of 1938 to regulate production, but one of the five Commissioners has recently altered his point of view. To clarify the Commission's authority, and to maintain the attitude heretofore taken, bills known as the Lyle and Harris bills have been introduced into the House, and more recently Bill No. 1498 has been introduced into the Senate by Senator Kerr, of Oklahoma, on behalf of himself and Senator Thomas, also of Oklahoma.

If peacetime natural gas production and its sales can be controlled by the Federal Power Commission, then it is argued that all mineral resources could likewise be controlled, which is the occasion of the present alarm by some of the mining group.

We have asked one more conversant with the situation than are we to prepare a statement which we are glad to quote as follows:

"Americans who believe that private development of our mineral resources will continue to produce the greatest benefits for the country should be interested in legislation introduced in the 81st Congress. The proposed law, known as the "Kerr Natural Gas Bill," would amend the Natural Gas Act of 1938 by more clearly defining the powers of the Federal Power Commission over the production and gathering of natural gas.

"Briefly, the National Gas Act of 1938 gave the Federal Power Commission power to regulate *interstate* transportation of natural gas and its sale in *interstate* commerce for resale. The Act did *not* grant power to regulate production or gathering. In turn this was long believed to exclude power to regulate the prices charged by producers and gatherers.

"In recent years, however, the jurisdiction of the Commission has gradually been extended, until now a majority of the Commission's members assert that it should have full control over the prices charged by producers and gatherers of natural gas.

"What does this mean? In operations over which it has jurisdiction, the Commission never allows, under court-approved principles, a return greater than 6½ per cent of the original cost of the properties involved. This rule, if applied to an oil field which produces both oil and gas—and that includes most oil fields in this country—could prevent any return at all from the production of casinghead gas, if the producer has already received the allowed return from his oil production. Furthermore, in the case of a purely gas-producing field, returns from the sale of gas might be zero if the producer had already recovered the original cost of his property. Also, since the base figure on which a fixed return is allowed is the original cost of a property, no allowance is made for the enhanced value of properties due to new discoveries.

"Under such circumstances it is obvious that the inducements for a property owner to invest in facilities for the production and gathering of natural gas are seriously impaired.

"The Kerr Natural Gas Bill, realizing that Federal regulation of the prices at which natural gas may be sold in the field by producers and gatherers would inhibit the development of this natural resource, and therefore is against the public interest, specifically prohibits such price regulation. Because of the diversity of opinion as to the powers granted by the 1938 Act, such clarification seems desirable.

"The Federal Power Commission has also claimed for itself the power to control the uses to which natural gas shall be put, and to prohibit the sale of natural gas in competition with other fuels in certain areas. It would seem, however, that the end use of any product should be governed by prices, utility, and the desires of the purchaser, and not by government edict.

"Except in time of war, it is not the function of government to control the end use of natural gas any more than it is its function to control the use of lumber, corn, or cotton; coal, lead, or iron. Certainly control over the end use of one commodity could lead to control over the end use of others.

"Eventually, unless restricted by the Kerr Natural Gas Bill or similar legislation, the Federal Power Commission could extend its jurisdiction so as to drive private enterprise out of the natural gas business. Once that is accomplished, it would be a short step to Government control of oil, coal, copper, and other natural resources—with Federal ownership or state socialism as an ultimate result.

"The bill now pending would merely amend the Natural Gas Act of 1938 to limit the Federal Power Commission to the powers originally thought to be conferred upon it.

"It is entirely appropriate for members of the AIME to advise their Senators and Representatives about their views and wishes on this subject."

Jobs Wanted

Our available space is again confined to one page this month, but we do want to get in a word about the plight of the June engineering graduates. Over 1200 of them have registered with the Engineering Societies Personnel Service for jobs as this is written. Of these, 105 are chemicals, 100 civils, 345 electricals, 625 mechanicals, 55 mining and metallurgical, and 40 geology and physics. The Service has approached many large potential employers but is receiving only two or three requests a week for men.

If an employer who reads this can possibly place one of these budding engineers, even if a less well-trained man must be displaced in the process, the investment should prove a good one. Give a man a job, train him, and have him ready when you need him.

File of Key Engineering Personnel Object of EJC Questionnaire

To provide source material for a who's who in engineering research, development, and other scientific operations for use by the National Military Establishment, the Engineers Joint Council acting through The American Society of Mechanical Engineers has accepted the task of providing the Office of Naval Research with the names, addresses, ages, and details of professional and scientific qualifications of 100,000 key engineers in all branches of American engineering.

The source file of key engineering personnel thus obtained will provide a valuable tool for solving a variety of technical personnel problems. Its use will diminish disturbances of the national economy, organization of industry, and the personal welfare of engineers, and provide a means by which national resources of technical personnel can be ascertained. The file will also point up weak spots which should be strengthened by education, training, and other means. As a national asset the body of facts will be available to private industrial, educational, and professional-society planning groups, and for other legitimate purposes.

A four-page questionnaire has been mailed to 100,000 engineers holding the grade of member or higher in eighteen national professional engineering societies. The returns will be collected by the ASME and turned over to the Office of Naval Research of the National Military Establishment for classification.

The project is the result of a conference held in Washington, D. C., last fall attended by EJC representatives and many other engineering agencies, at which was discussed the need for a list of 25,000 key engineers working in research, development, and other scientific projects who could be called in on a full or part-time basis to work on the broad scientific programs of the National Military Establishment. The task of collecting personal and professional data fell to the EJC as the largest joint agency of the engineering profession. ASME assumed administration of the project as contracting agent of the EJC.

The Engineers Joint Council points out that this is not just another questionnaire, but one sent to engineers selected from the upper echelon of the profession. The data sought is not intended for general government use but will go directly to the engineering agencies of the National Military Establishment, who will make direct use of it. As the questionnaire will provide the key to opportunity to professional and patriotic service, engineers selected to receive it are urged to give it serious attention and to answer all questions fully.

Calendar of Coming Meetings

AUGUST

4 Reno Branch, Nevada Section, AIME.
10 El Paso Metals Section, AIME.
16 Gulf Coast Section, AIME.
22 Alaska Section, AIME.

SEPTEMBER

25-28 American Mining Congress, Metal Mining Convention, Hotel Davenport, Spokane, Wash.
25-28 Mid-year Meeting, AIME, Neil House, Columbus, Ohio.
29-30 ASME, fall meeting, Erie, Pa.
29-Oct. 1 Colorado School of Mines, 75th anniversary celebration.
Sept. 30-Oct. 1 Southern Ohio Section of Open Hearth Committee, AIME, fall meeting, Deshler-Wallick Hotel, Columbus.

OCTOBER

3-4 National Assoc. of Corrosion Engineers, Adolphus Hotel, Dallas.
5-7 Petroleum Branch, AIME, fall meeting, Plaza Hotel, San Antonio, Texas.
13-14 Texas Mid-Continent Oil and Gas Association, annual meeting, Rice Hotel, Houston.
14 Eastern Section, Open Hearth Committee, Iron and Steel Division, annual all-day fall meeting, Warwick Hotel, Philadelphia.
14 Southwestern Section, Open Hearth Committee, Iron and Steel Division, Kansas City, Mo.
17-19 Institute of Metals Division, AIME, fall meeting, Allerton Hotel, Cleveland.
17-21 National Metal Congress and National Metal Exposition, Public Auditorium, Cleveland, Ohio.
17-21 American Society for Metals, annual meeting, Cleveland, Ohio.
17-21 American Welding Society, annual meeting, Cleveland, Ohio.
17-23 AIEE, 1949 Mid-West meeting, Netherland Plaza, Cincinnati.
19-20 Society for Non-Destructive Testing, Cleveland, Ohio.
20-21 Petroleum Branch, AIME, Elks Club, Los Angeles.
24-28 Thirty-seventh National Safety Congress and Exposition, Chicago.
26-27 Joint Fuels Conference, ASME-AIME, French Lick Springs Hotel, French Lick, Ind.
28 Pittsburgh Section of Open Hearth Committee and Pittsburgh Section, AIME, annual fall meeting, William Penn Hotel, Pittsburgh.
28-29 ECPD, annual meeting, Edgewater Beach Hotel, Chicago.

NOVEMBER

1-5 Pacific Chemical Exposition, California Section, American Chemical Society, San Francisco Civic Auditorium.
2-4 American Society of Civil Engineers, fall meeting, Washington, D. C.
7-10 AIChE, annual meeting, Pittsburgh, Pa.
7-12 International congress on tunnel driving in rock formation, organized by the Societe de l'Industrie Minerale. Information on meeting available from French Mining Mission, 1322 18th St., N. W., Washington, D. C.
9-11 Industrial Minerals Division, AIME, Tampa, Fla.
10-14 ASTM, first Pacific area national meeting, San Francisco, Calif.
12-14 Geological Society of America, annual meeting, Hotel Cortez, El Paso.
16-18 Industrial Hygiene Foundation, 14th annual meeting, Mellon Institute, Pittsburgh.

DECEMBER

7 American Mining Congress, Annual Business Meeting, New York City.
8-10 Seventh Annual Conference, Electric Furnaces Steel Committee, Iron and Steel Division, AIME, Hotel William Penn, Pittsburgh.

JANUARY 1950

18-20 American Society of Civil Engineers, annual meeting, New York.
30 AIEE, winter meeting, New York.

Do Not Send Cash to the AIME

Occasionally members send small amount of cash to the Institute in payment for publications, and sometimes even pay their dues with five and ten dollar bills. Ordinarily such sums arrive safely, and are properly credited, but occasionally such remittances are never received. The Institute wishes again to emphasize that it cannot be responsible in such instances, and urges members to use checks or money orders even for small amounts.

Approaching Prospects for Membership

Among the duties and privileges of all AIME members interested in building up their professional society is the securing of new members. If every member would hold himself responsible for securing just one new member, the growth and prosperity of the AIME would show a startling upturn. Application blanks will gladly be forwarded on receipt of a postal card request. Many blanks are now outstanding, and it should be made clear, in approaching prospects, that the dues for the next three years will be $5 higher for Members and Associate Members, and $2 higher for Junior Members, than is stated on the application blanks.

News of AIME Members

Lew Adamec is at present section superintendent of Cie. Aramayo de Mines en Bolivie, Casilla 674, La Paz, Bolivia. He previously held the post of level foreman for the Braden Copper Co. at Rancagua, Chile.

D. L. Anderson is now addressed at Empresa Minera de Nicaragua, Apartado 195, Managua, Nicaragua.

George Ross Bancroft, Canadian consulting mining engineer, has now changed his address from Vancouver, B. C., to Okanagan Mission, B. C., which will be his future home and address.

Ralph C. Beerbower and **Ralph C. Beerbower, Jr.**, father and son, appeared in the July News all mixed up; the fault was ours. Mr. Beerbower, Sr., is district manager of the Goodman Mfg. Co. Mr. Beerbower, Jr., after graduating from Penn State in 1948, joined the H. C. Frick Coke Co. and was recently assigned to the post of assistant to the superintendent at Leisenring No. 3 Mine, West Leisenring, Pa. We apologize to both gentlemen.

Arthur H. Bunker

Arthur H. Bunker resigned as a general partner of Lehman Brothers on June 30 to become president of the Climax Molybdenum Co. At present a director of the American Metal Co., the Firth Steel and Carbide Corp., and Climax Molybdenum, Mr. Bunker was the founder in 1923 and first president of the U. S. Vanadium Corp. During the recent war he served with the OPM and WPB, becoming chief of staff of the latter.

T. W. Lippert

T. W. Lippert has been appointed manager of publications by the AIME. In this capacity he will also act as editor of the *Journal of Metals*. Mr. Lippert is a graduate of Carnegie Institute of Technology, with a B.S. and M.S. in physics, and for the past sixteen years has been associated with *Iron Age*, serving in latter years as directing editor of that publication, and as a director of the parent Chilton Co.

J. W. Bostick plans to remain at 6028 Folsom Drive, La Jolla, Calif., until next November. His office is still at 3116 Main St., Dallas, Texas, where mail or calls are promptly forwarded by his secretary. Being a University of California man, he has met many old friends and college mates there, in Los Angeles, and in San Francisco.

Arthur E. Buller, who was formerly resident geologist for the Kelowna Exploration Co. at the Nickel Plate Mine, Hedley, B. C., has accepted the post of chief geologist for the Pend Oreille Mines and Metals Co. at Metaline Falls, Wash.

R. W. Crosby is on the staff of the Benguet Consolidated Mining Co., Baguio, P. I. He was previously employed by Marsman and Co. in Manila.

Douglas MacDonald Dunbar, who had been in Chuquicamata with the Chile Exploration Co., has been made assistant to the executive vice-president of that Company. He is now at Company headquarters located at 25 Broadway, New York City 4.

Howells Frechette has been elected vice-president of the American Ceramic Society. Former chief of the industrial minerals division of the Bureau of Mines, Ottawa, he is the first Canadian to hold this post.

Cy W. Greenhalgh, who was research metallurgist for the Remington Arms Co., Bridgeport, Conn., has recently joined the staff of the Old Town Co., in Old Town, Maine, as general manager. He resides at 144 College Rd., Orono.

Jean McCallum retired on May 1 from his service with the National Lead Co., completing a period of almost exactly 31 years. He became connected with National as assistant superintendent of the River Smelting and Refining Co.,

Jean McCallum

Florence, Colo., in 1918, where he handled all phases of the operation and assisted in perfecting the metallurgy at many points. When this plant ceased operations, he was transferred to Collinsville, Ill., as superintendent of the lead smelter of the St. Louis Smelting and Refining Co., from which point he moved to St. Louis as research and development metallurgist for the National Lead branch and Titanium Division. Following five years as plant superintendent of the whitelead and oxide plant, he was made vice-president of the St. Louis Smelting and Refining Co. and manager of the St. Louis Smelting and Refining Works, division of National Lead, which post he held until his retirement. In 1947 he was also made general manager of the Texas Mining and Smelting Division at Laredo, Texas.

Curtis L. Wilson, dean of the School of Mines and Metallurgy, University of Missouri, watches Andrew Fletcher receive an honorary degree of Doctor of Engineering from F. A. Middlebush, president of the University, at the commencement exercises in Rolla, at which Dr. Fletcher delivered the address.

Paul W. Fairchild received an M.S. degree in geology from the University of Kansas in February and is now employed as a junior geologist with the Shell Oil Co. exploration department in a one-year training program.

E. W. Hunt has been appointed superintendent of mines with the Hudson Bay Mining and Smelting Co., Flin Flon, Man.

E. J. Carlyle

E. J. Carlyle officially retires as executive director and secretary-treasurer of the Canadian Institute of Mining and Metallurgy on Sept. 30. He was born in Woodstock, Ont., in 1877 and graduated in mining engineering in 1904 from McGill University. Upon graduation, he received early professional training at Anaconda, Mont., and Arizona. He held various important posts in several foreign countries, as well as Canada, such as for the Sissert Corp. in Russia for whom he designed a smelter and refinery at Salt Lake City. Before his appointment as secretary-treasurer of the Canadian Institute in 1931, he was superintendent of the American Smelting and Refining Co. in Peru. He was made executive director and secretary-treasurer in January 1947.

L. H. Lange, vice-president and consulting metallurgist of the Galigher Co., Salt Lake City, sailed with his wife from New York City on June 15 on the "Robin Hood" for Southwest Africa where he plans to spend several months at Tsumeb. After leaving Tsumeb, time will be spent visiting Johannesburg and the copper properties in Northern Rhodesia before returning to the States.

Geoffrey B. Leech, who had been attending Princeton University, is now working for the Geological Survey of Canada, Ottawa, Ont.

William F. Machonis is employed as a service engineer by Combustion Engineering Superheater Inc., Philadelphia, Pa.

Willam B. MacPhee, formerly of the Minerals Engineering Co. of Los Angeles, is at present with the Castrovirreyna Metal Mines Co., Casilla 101, Santa Ines, Huancavelica, Peru.

Howard Maidment, who had been a student at the University of British Columbia, can be reached at the Polaris-Taku Mining Co., Tulsequah, B. C.

Frederic H. Main, upon leaving Columbia University where he was a lecturer in economic geology during the absence of C. H. **Behre, Jr.**, is now on the staff of the Bertha minerals division of the New Jersey Zinc Co. at Austinville, Va.

W. B. Mather, chairman of the division of mineral technology of the Southwest Research Institute, San Antonio, Texas, before June 1 was with the Midwest Research Institute in Kansas City, Mo.

Richard D. Mayne, manager of the Hannibal, Mo., plant of the Universal Atlas Cement Co., a U. S. Steel subsidiary, became manager of the company's plant at Hudson, N. Y., on June 1. A 1930 graduate of Carnegie Institute of Technology, he joined Universal Atlas in Chicago in 1937 as an industrial engineer. After serving the cement company in various capacities he was appointed assistant plant manager at Hannibal in 1943 and three years later became plant manager.

John B. McIntyre, formerly a student at Oregon State College, is at present working for the U. S. National Museum, Room 331, Washington 25, D. C.

John F. Meissner is president of John F. Meissner Engineers, Inc., 308 West Washington St., Chicago 6. He had been associated with Robins Conveyors, Inc., also of Chicago.

William C. Miller, who was a student at the University of California at Berkeley, is at present working as an assistant mining engineer for the Anaconda Copper Co. He can be reached at 615 W. Park, Butte, Mont.

Robert W. Michael, specialist in the design of diamond recovery equipment, is now engaged with the New York Engineering Co. as consulting engineer in the fabrication of the first mechanical diamond washing unit to be installed in the new Venezuelan alluvial fields. The plant will contain many unique mechanical features, and is to be constructed so that it can be demounted for air transport. The flow sheet will include the latest developments in diamond recovery practice. Mr. Michael may be reached in care of the New York Engineering Co., 75 West St., New York 6.

Frank R. Milliken, assistant manager, Titanium Division, of the National Lead Co., can now be reached at 111 Broadway, New York City, having transferred from Tahawus, N. Y.

B. E. Mix has retired as plant engineer of the Utah Copper Co. at Garfield, Utah, and is now residing at 2767 Claybourne Circle, Salt Lake City.

E. S. Moore, head of the department of geological sciences, University of Toronto, and formerly dean of the school of mines, Pennsylvania State College, retired on June 30 with the title Professor Emeritus after 27 years on the staff.

Jack E. Morris, research engineer of the general technical department of Jones & Laughlin Steel Corp., although still in the same department, can now be reached care of the Company, Jones & Laughlin Bldg., 3rd and Ross St., Pittsburgh 20. He had previously been at Star Lake, N. Y.

R. G. K. Morrison is returning to Canada, and on Sept. 1 will take over the duties of professor of mining engineering at McGill University. Mr. Morrison was a director of the Mysore Gold Mining Co., Ltd., and a managing director of John Taylor & Sons (India), Ltd. The latter company, being an off-shoot of the old London firm of John Taylor & Sons, was formed to take over interests in India, including the management of the four operating companies of the Kolar gold field.

Eric A. Rudd, having resigned from the position of chief geologist of the Broken Hill Pty. Co., of Melbourne, Australia, has taken the chair of mining and economic geology at the University of Adelaide. This is the first university department of mining and economic geology in Australia; Professor Rudd hopes that it will become the center for teaching mining geology on that continent from now on. He can be addressed at the University, North Terrace, Adelaide.

William F. Jahn

William F. Jahn recently returned to the United States from Argentina after eight years consulting work in that country and is now enjoying a vacation on his ranch at Stark, Missoula County, Mont.

Frank E. Siegfreid graduated from the Colorado School of Mines in May and is now employed by Telluride Mines Inc., Telluride, Colo.

Robert C. Stanley, chairman and president of the International Nickel Co. of Canada, delivered the address at the spring convocation held on May 21 at Queen's University, Kingston, Ont. He was the recipient of the honorary degree of LL.D. along with **W. B. Timm,** who retired recently as director of mines, forests and scientific services branch, Department of Mines and Resources; and **Stanley N. Graham,** former head of the department of mining at Queen's.

Waters, Kudlich, Otto, James, Reinhardt, backed by Romischer, Oehler, Jenkins, Boyd, McGleserey, Reed, Petzold, and Thomas pose at the Marvine colliery of the Hudson Coal Co., Scranton, Pa., when James Boyd, director of the Bureau of Mines, made his first visit to the anthracite northern coal field. He was taken on a tour by officials of the Glen Alden Coal Co., Lehigh Coal Co., and the Hudson Coal Co. At the evening banquet in Scranton, Dr. Boyd gave a talk on general economics of the metals and minerals of this country.

Blakemore E. Thomas, of the California Institute of Technology, has been appointed assistant professor of geology at the University of Kansas, Lawrence. Dr. Thomas will teach work in petrography and economic geology.

H. W. Thoms, geologist with the Creole Petroleum Corp., Caracas, Venezuela, will be on vacation in the States from July 1 to Aug. 19. His address while here will be 242 Pacific St., Bakersfield, Calif.

Robert E. Tally, Jr., can be reached care of Eastern Mining and Metals Co., Kuala Dungun, Trengganu, Malaya. He had been with Foley Bros., Inc., at Pasadena, Calif.

Robert Van Nostrand has earned his degree of Master of Science in physics at the Missouri School of Mines and is now working for his doctorate in geology at the University of North Carolina.

C. E. Visel, formerly with the Cia. Minera Nacional at Chihuahua, is now on the staff of Negociacion Minera, Sta. Maria de la Paz y Anexas, Matehuala, S.L.P., Mexico.

Dooley P. Wheeler, Jr., geologist, is now with the American Metal Co.; his address is 232 Judge Bldg., Salt Lake City.

Robert A. Willoughby joined the staff of the Yucca Mining and Milling Co., Box 67, Yucca, Ariz. He had been with the Union Sugar Co. at Betteravia, Calif.

William Wraith, Jr., recently changed his job from the Cananea Consolidated Copper Co. in Sonora, Mexico, to the Chile Exploration Co. in Chuquicamata, Chile.

James H. Wren, after covering a number of Central and South American mining examinations, is back in the United States. He has taken the operational consulting account of the Sunshine Gold Mining Company's Redding, Calif., properties, which are about to be put under production. His address is 4317 D St., Sacramento, Calif.

Charles E. Wyndham is now employed as a mining engineer by the Bureau of Mines and can be addressed care of the Bureau, Box L, University, Ala. He previously worked for the Standard Lime and Stone Co., Kimballton, Va.

Harry J. Wolf

Harry J. Wolf, of Behre, Dolbear and Co., was in Mexico recently inspecting mineral deposits in the state of Chihuahua.

• In the Metals Divisions

Glenn L. Allen, previously associated with L. W. Kelley of Park Ridge, Ill., is now employed by the Reynolds Metals Co., 3rd and Grace Sts., Richmond, Va.

Jay W. Fredrickson

Jay W. Fredrickson, formerly at the University of Utah, has joined the staff of the School of Mineral Industries, Pennsylvania State College, as chief of the division of metallurgy.

Samuel E. Atkins, president of the S. E. Atkins Co., has announced the incorporation of Atkins-Walker Co. Offices will continue at 612 Alworth Bldg., Duluth, Minn. Drilling operations which have been carried on for 29 years by S. E. Atkins Co. were turned over to the new company on May 2.

Francis O. Case, former assistant to the vice-president of the Anaconda Copper Mining Co., has been made a vice-president of the Company.

Raf De Bie has finished his studies at Carnegie Tech and received a master's degree. He is touring the country to see something of our industries and will return to Belgium early in August to start working in the research department of the Trefileries Leon Bekaert at Zwevegem.

Robert D. Fitzgerald has taken the job of assistant metallographer with the Aluminum Co. of America at New Kensington, Pa. His home address in New Kensington is 309 Charles Ave.

Arnold W. Greenius is now attached to the department of research and development of the Dominion Steel and Coal Corp. of Sydney, N. S.

Max Hansen has left his associate professorship at the Illinois Institute of Technology to take a position with the Armour Research Foundation which is located on the IIT campus. Members of the AIME Student Chapter at the school presented Dr. Hansen, who was one of the students' favorite instructors, with a going-away present. The professor had come to the U. S. A. but two years ago, and his departure from the IIT faculty is considered as "great loss to both the department and the students."

John C. Kinnear on June 6 delivered the commencement address to 227 graduates of the University of Nevada. Mr. Kinnear, vice-president of the Kennecott Copper Corp. of New York City, was the recipient of an honorary degree of doctor of laws. In addition to his mining activities, he is vice-president of the Nevada Northern Railway and of the Ely National Bank. **Jay A. Carpenter,** director of the Mackay School of Mines of the University of Nevada, was awarded the honorary degree of doctor of science.

Fritz V. Lenel has been made associate professor in the department of metallurgical engineering at Rensselaer Polytechnic Institute. He earned his doctorate at the University of Heidelberg in 1931, and did graduate work at the University of Goettingen until 1933. He joined the Rensselaer staff in 1946 after ten years with the Moraine products division of General Motors.

John W. Axelson

John W. Axelson recently received a Ph.D. degree from the University of Minnesota where he majored in chemical engineering and minored in metallography. He is now employed as a research engineer doing research and development work on metallic gaskets in the research center of the Johns-Manville Corp., Manville, N. J.

Robert F. Mehl, director of the metals research laboratory and head of the metallurgical engineering department at Carnegie Institute of Technology, Pittsburgh, flew to Brazil on June 29 to be a guest for about four weeks of the Research Institute of the State of Sao Paulo at their fiftieth anniversary. Dr. Mehl will also take part in the dedication of new laboratories at the Cidade Universitaria, and will attend the fifth anniversary of the Brazilian Society of Metals, which he helped organize. He lectured in Brazil in 1943 at the request of Nelson A. Rockefeller, co-ordinator of Inter-American Affairs. Dr. Mehl is a permanent member of the faculty of the University of Sao Paulo, from which he received an honorary doctor's degree.

Norman L. Mochel, manager of metallurgical engineering for the Westinghouse Electric Corp., has been elected a member of the board of the American Society for Testing Materials for a three-year term. First employed in the inspection department of the then Westinghouse Machine Co., he later was responsible for the work of testing materials. During the first World War he served overseas in the Corps of Engineers. For many years he has been in his present position, an authority on the materials going into the turbine and other forms of power generation.

George F. Newton is a trainee with the Steel Co. of Canada, Hamilton, Ont. His mail goes to 1077½ Beach Blvd., Hamilton.

Antonio C. Oliveira is now assistant engineer of the foundry department of Laminação Nacional de Metais, and his mailing address is Rua Heliotropos, 245 (Vila Mariana), São Paulo, Brazil. He had been associated with the Cia. Siderurgica Nacional at Volta Redonda.

Bruce Pierce was married to Helen Decker on June 18 at a ceremony at the Metropolitan Methodist Church in Washington, D. C. The groom, who is in the ordnance department of the regular army as Lieutenant Colonel, is at present taking a two-year graduate course in engineering at Cornell University. The bride is a native of Washington, D. C., and a graduate of Wilson Teachers College. The couple will reside at R. D. 4, Ithaca, N. Y.

Adolph O. Schaefer, assistant to the executive vice-president of the Midvale Co. of Philadelphia, has been appointed to the research committee of Franklin Institute. A native of Pennsylvania, residing in Broad Ax, Pa., he graduated from the University of Pennsylvania in 1922 with a B.S. in chemical engineering. He joined Midvale as a research assistant shortly after graduation and became engineer of tests in 1937 and executive metallurgical engineer in 1944. He held many important posts during the last war, one of which was that of chairman of the gun development group in the Metallurgical Advisory Board.

Howard Peters

Howard Peters was appointed as project engineer for the rollpin division of the Elastic Stop Nut Corp. of America, Union, N. J. He was previously a vice-president of the Mid-Continent Metal Products Co. of Chicago, and during his employment with that Company developed the process and machinery for producing the rollpin fastener. Mr. Peters is a graduate of Purdue University with a B.S. and M.S. in engineering, and was a member of the staff of Battelle Memorial Institute at Columbus. He served in the U. S. Army during the recent war.

Herbert A. Robinson has the job of research metallurgist at Battelle Memorial Institute, Columbus 1, Ohio. He had been with the American Smelting and Refining Co. at Barber, N. J.

Robert Maddin recently accepted the post of assistant professor of mechanical engineering at Johns Hopkins University. He was formerly a research fellow in metallurgy at Yale.

George B. Waterhouse, emeritus professor of metallurgy at MIT, and consulting metallurgist, was awarded an honorary Doctor of Engineering at the recent Nova Scotia Technical College convocation in Halifax. Dr. Waterhouse delivered the convocation address.

Clyde Williams, director of Battelle Memorial Institute, will give the introductory paper on the conservation of iron and steel in production at the United Nations' Scientific Conference on the Conservation and Utilization of Resources on Aug. 25 at Lake Success, N. Y. **John D. Sullivan,** assistant director of Battelle, is scheduled to present a paper on the new processes for the utilization of low-grade ores five days later. The conference is to be scientific rather than policy-making, according to the UN's Economic and Social Council. It will have no power to bind governments and it will not formulate recommendations to them. The practical application of science to resource management and human use will be the primary concern. During the conference papers will be presented by approximately 450 scientists, engineers, and other experts from many countries.

In recognition of significant contributions in the fields of science and public service, the honorary degree of Doctor of Engineering was recently conferred on Mr. Williams by the Michigan College of Mining and Technology. Prior to the presentation, he delivered the commencement address to over 500 candidates of the Michigan graduating class.

Lincoln T. Work has become associated with Alan R. Lukens at the Powdered Material Research Laboratories, 11 Windsor Street, Cambridge 39, Mass. With the coming of Dr. Work, the organization's field of utility will be widened to undertake work on abrasives, refractories, metal powders, cements, and dusts. He is an authority on grinding, classification, and filtration. Dr. Work

Lincoln T. Work

had been director of research and development for the Metal and Thermit Corp. for the past nine years. He was on the staff of the chemical engineering department at Columbia University before his affiliation with the Metal and Thermit Corp., and was at that time well known in consulting engineering.

• In Petroleum Circles

Floyd D. Aaring, former University of Oklahoma student, is a petroleum engineer with the Canadian Gulf Oil Co., P. O. Box B-4, Pincher Creek, Alta.

T. H. Acres is on the staff of the Barnsdall Oil Co., Box 367, Newhall, Calif.

George P. Alden, formerly with the U. S. Geological Survey, has the job of petroleum production engineer with the Barnsdall Oil Co., P. O. Box 2039, Tulsa, Okla.

Arnold D. Arnaut graduated in June from the University of Wisconsin's metallurgy department and is employed by Sylvania Electric Products, 35-22 Linden Place, Flushing, L. I., N. Y.

Leo R. Brammer, Jr., who has been studying at the University of Tulsa, is an engineer with the Dempsey Pump Co., Box 3098, Tulsa, Okla.

E. J. Brook is president of the McElroy Ranch Co. of Fort Worth, Texas. He had been with the Franco Wyoming Oil Co.

Robert E. Brooks is working for the Skelly Oil Co. in the Southern division on production. His address for mail is P. O. Box 224, Wynnewood, Okla.

W. T. Cardwell, Jr., formerly with the Standard Oil Co. of California, is senior research engineer for the California Research Corp., La Habra, Calif.

Campbell M. Carothers, district petroleum engineer for the Humble Oil and Refining Co., is addressed at Rt. 1, Raymondville, Texas.

A. L. Carpenter has the post of junior petroleum engineer in the production department of the Humble Oil and Refining Co., Bay City, Texas.

Aaron Cawley, who has been studying at the University of Texas, is now working as engineering trainee for the Honolulu Oil Corp. He can be addressed at Box 1079, Denver City, Texas.

John H. Chain has a job with the American Republics Corp., Box 547, Artesia, N. Mex. He was a student at the University of Texas.

Thomas E. Covington is an engineer trainee with the Tidewater Associated Oil Co., Box 12, Venice, La.

Joseph E. Barthelemy, Jr., who attended the University of St. Louis, is now employed by the Western Geophysical Co., P. O. Box 103, Lac La Biche, Alta.

Rex M. Everett is working for the Magnolia Petroleum Co. in Kermit, Texas. He had been a student at the University of Oklahoma.

Stewart H. Folk has resigned his post at Baylor University in order to accept an assignment with DeGolyer and MacNaughton, consulting geologists and engineers, as their representative in Mexico. His address is now in care of Petroleos Mexicanos, Depto. Exploracion,

XUM

Rm. 208, Avenida Juarez 95, Mexico City, D. F.

Thomas E. Goebel is in Maracaibo, Venezuela, working for the Cole Drilling Co.

Harold C. Kidd, who attended the University of Tulsa, is on the staff of the Creole Petroleum Corp., Apt. 172, (Laquinillas), Maracaibo, Venezuela.

J. B. Ladd received a B.S. degree in petroleum engineering from the University of Kansas in June and is now employed by The Texas Co. in Cut Bank, Mont.

Kenneth H. Larson, a former student at the Montana School of Mines, is now a junior computer of the Stanolind Oil and Gas Co. His mail address is 1145 Monroe, c/o Stanolind Oil and Gas Co., Seis Party 14, Alexandria, La.

Clinton C. Lieffers, formerly on the staff of the Texas Amerada Petroleum Corp., is now working in the geology department of the Creole Petroleum Corp., Apt. 889, Caracas, Venezuela.

William G. Locke is tar products sales cadet in the tar products division of the Koppers Co., Pittsburgh.

Arthur F. Malicoat is a paleontologist with the Gulf Oil Corp. He receives his mail at his home, 1302 S. 49th St., Temple, Texas.

W. J. Mannas writes that his mail should be addressed to the Pure Oil Co., 35 E. Wacker Drive, Chicago.

William J. McPherson has changed his address from Stanford University to care of Imperial Oil Ltd., 300 9th Ave. W., Calgary, Alta.

Ralph J. Poth has taken a job with the Union Sulphur Co., with Box 511, Sulphur, La., for an address.

Dayton E. Ramsey recently joined the staff of Dowell, Incorporated, Kermit, Texas. He is a former student of Texas A&M.

P. H. Rumsey is addressed in care of the Iraq Petroleum Co., Ltd., (Exploration), Tripoli, the Lebanon. He took the job of junior geologist with the company over a year ago.

H. J. Schroeder has gone to Talara, Peru, as petroleum reservoir engineer with the International Petroleum Co. He had been in the Baroid Sales Division of the National Lead Co. at Houston.

Richard G. Stephens, who studied at the University of Texas at Austin, and was formerly employed by The Texas Co. as a trainee, is at present on active duty in the U. S. Navy. His mail will reach him at 3206 King's Rd., Dallas, Texas.

John F. Stickel, Jr., is a geological engineer with the Wesley, Beford and Stickel Co. of Detroit, Mich.

Todd C. Storer has been transferred by the Stanolind Oil and Gas Co. to Bishop, Texas, where he is field engineer in the Luby area.

Marshall C. Turner is production superintendent for the Continental Consolidated Corp., Long Beach, Calif. He had been with the Union Oil Co.

William E. Topley, a Montana School of Mines student, is now geologist for the Stanolind Oil & Gas Co., 336 8th Ave. West, Calgary, Alta.

Necrology

Date Elected	Name	Date of Death
1918	Paul Armitage	June 28, 1949
1936	William A. Bauerus	Feb. 23, 1949
1936	William Bell	1948
1945	W. A. Bishop	June 10, 1949
1942	T. C. Botterill	June 15, 1949
1947	Toro Flygare	November, 1947
1918	Walter Miller	Unknown
1916	Clarence J. Peterson	May 8, 1948
1904	E. A. Cappelen Smith	June 25, 1949

Proposed for Membership

Total AIME membership on June 30, 1949, was 15,759; in addition 4424 Student Associates were enrolled.

ADMISSIONS COMMITTEE

James L. Head, Chairman; Albert J. Phillips, Vice-Chairman; George B. Carless, T. B. Counselman, Ivan A. Given, George C. Heikes, Richard D. Mollison, and Philip D. Wilson.

Institute members are urged to review this list as soon as the issue is received and immediately to wire the Secretary's office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate; F, Junior Foreign Affiliate.

ALABAMA

Birmingham — AUSTIN, WILLIAM WYATT, JR. (C/S—J-M). Research metallurgist, Southern Research Institute. SANDELIN, ROBERT WILLIAM. (C/S—J-M). Chief metallurgist, Stockham Values & Fittings, Inc.

ARIZONA

Morenci—BACON, FREDERICK MASON. (AM). Flotation operator, Phelps-Dodge Corp.

Phoenix—POPPINO, CARL ALBERT. (AM). Sales engineer, General Electric Co.

ARKANSAS

Magnolia—ALGER, ROBERT PERDUE. (C/S—J-M). District engineer, Schlumberger Well Surveying Corp.

CALIFORNIA

Alhambra — JAMISON, PAUL HUGUS, JR. (C/S—J-M). Sales promotion engineer, Kobe, Inc. STICK, JOHN C., JR. (C/S—J-M). Chief electrical engineer, Lane-Wells Co.

Bakersfield — CARRICK, HARRY HALL, JR. (C/S—J-M). Resident engineer, General Petroleum Corp. MANGOLD, ROBERT P. (C/S—J-M). Petroleum engineer, Independent Exploration Co. PERRY, CLYDE WILBERT. (C/S—J-M). Division petroleum engineer, Richfield Oil Corp. YAMAMOTO, ROY HIDEO. (R,C/S—S-J). Assistant engineer, bottom-hole lab., Core Laboratories, Inc.

Berkeley—PARKER, EARL R. (C/S—J-AM). Associate professor of metallurgy, University of California. WICKETT, WALTON AMES. (R,C/S—S-M). Western sales manager, Tracerlab, Inc.

Huntington Park — ANDERSON, JAMES LOUIS. (M). Chief engineer, Conveyor Co. WISEMAN, KENNETH WILLIAM. (J). Trainee-petroleum engineering, Socony-Vacuum Oil Co.

La Habra — SWENY, JOHN WILLIAM. (C/S—J-M). Associate research engineer, California Research Corp.

Lompoc — GRIMM, KENNETH E. (C/S—J-M). Paleontologist, Johns-Manville Products Corp.

Long Beach — CARRIEL, JAMES TURNER. (C/S—J-M). Petroleum engineer, Southwest Exploration Co. HILTY, GEORGE CLIFFORD. (C/S—J-M). Petroleum engineer, Long Beach Harbor Dept. INSKEEP, WILLIAM DOLMAN. (M). Drilling contractor.

Los Angeles — BEAL, CARLTON. (C/S—J-AM). Consulting petroleum engineer. LESTER, GRAHAM WALLACE. (J). Engineer's assistant, Union Oil Co. of California. MASON, HAMILTON. (C/S—J-M). Service engineer, North American Aviation, Inc.

Orcutt—HILTON, ALFRED GEORGE. (M). Division drilling foreman, Union Oil Co. of California.

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Symposium on Western Phosphate Mining

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Foreword

By E. M. NORRIS,* Member AIME

Phosphate deposits are distributed widely over the earth's surface. Of the known areas of deposit, eight fields are of particular interest because of their vast reserves of high grade phosphatic material and the volume of their production that goes to the world's great consuming areas. These fields, in the order of their present economic importance, are situated respectively in Florida, French Morocco, Kola Peninsula of U.S.S.R., Tunisia and Algeria, Tennessee, Montana-Idaho-Wyoming-Utah, Ocean and Naru Islands in the Gilbert group, and in Egypt.

The Rocky Mountain deposits have a comparatively low ranking in the phosphate trade of today, but this field contains one of the largest known reserves of phosphatic material, associated with which are recoverable trace minerals, which are of increasing importance in our national economy. The first discoveries of phosphate rock in the western states were recorded in Cache[1] and Rich[2] counties in Utah, respectively in 1889 and 1897. Mining operations were begun on these deposits near Montpelier, Idaho, in 1906. During the latter year geological studies of the phosphate beds were started by Weeks and Ferrier of the U. S. Geological Survey. These studies have been continued to the present day by a distinguished roster of the Survey's technicians. As a result of this systematic exploratory program, the Survey has published a series of bulletins, papers, and maps, which provide the mining public with a comprehensive study of the exposed

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[1] References are at the end of this paper.

portions of the deposits. These publications have been of great value to those engaged in the commercial development of this field. To the hoary western miner, it is indeed a glimpse of Utopia, when sourdough and geologist, alike, have the well-thumbed pages of "Mansfield"[3] to guide their questing footsteps.

Production from the western field lagged for many years because of its remoteness from fertilizer markets and the resulting heavy transportation costs. During the years of 1919 and 1920, pilot plants were put into operation at Anaconda, Mont., for the manufacture respectively of sulphuric acid and high-analysis acid phosphate.

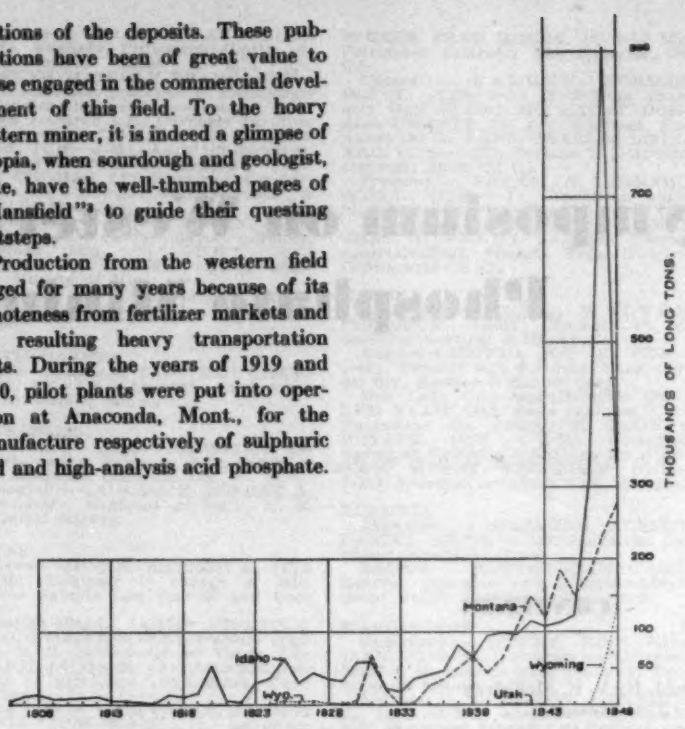

FIG 1—Idaho, Montana, Wyoming and Utah phosphate rock production, 1906 to 1948.

These plants were soon replaced by full scale operations. Fourteen years later enterprising agronomists of the western beet-sugar manufacturers discovered that proper application of phosphate fertilizers would increase sugar-beet yields, in the alkaline soils of the intermountain states, by approximately 16 pct. Thus a western fertilizer market was created and the advent of World War II caused a rapid expansion in production from the western deposits (Fig 1). Four major producers are now operating in this field and a number of properties are in various stages of development. It will be the purpose of this symposium to describe the most recent geological and mining developments.

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Geological Studies of the Western Phosphate Field*

By V. E. McKELVEY†

Introduction

The Phosphoria formation of the northwestern states presents a stimulating challenge to workers in every field of mineral technology. In addition to its large reserves of phosphate, the formation has been found to contain important amounts of fluorine and several metals, such as vanadium, nickel, and molybdenum. Because these minor metals in the Phosphoria do not have their usual glitter, concentrations of them are not easily discovered nor appraised. Like the phosphate itself, production of these metals very likely can be achieved only through use of costly chemical methods, for their mineralogy precludes treatment by the usual methods. In addition, in much of the area the containing rocks lie in complexly folded and faulted structures, which complicate mining and increase its cost. Yet the chain of research necessary to establish what elements are present; their geographic and stratigraphic distribution and reserves; the most suitable methods for recovering them; the combination of beds which, when mined and processed, will yield the greatest recovery of the most valuable elements; and the methods most adaptable to their profitable extraction may well lead to the development of an important mineral industry and may make possible a significant reduction in the cost of fertilizer—this in itself would be an achievement of momentous importance—and at the same time make the nation more self-sustaining in critical materials.

Although this research has great

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economic promise, most of the problems have proved so large, complex, and costly that a single company cannot afford to attack them, though all the larger companies have made important studies of certain technologic problems of immediate interest to them. Considering possible benefits to the nation, however, the odds against success are not too great for the public to risk, and it is therefore highly appropriate for the Federal Government to investigate the several types of problems named.

The U. S. Geological Survey's part in this investigation, in which a number of other agencies are participating, is to define what elements are present, where they occur, how much of each is present, in what structures the containing beds lie, and the origin of the rocks, and the elements contained in them. To achieve these various purposes satisfactorily we are examining previously unmapped areas in which the Phosphoria formation could occur, but has not yet been looked for or found; and we have begun to map, on a scale no smaller than 1:62,500, the geology and topography of all areas containing the Phosphoria formation; to measure, describe, and sample all beds of the phosphatic and shaly portion of the formation at scores of localities over the western field; and to analyze the samples for phosphate, fluorine, vanadium, minor metals, and, for certain beds, rock-forming constituents.

The mapping and sampling will (1) define the regional geologic structure, and (2) permit estimation of reserves (of phosphate, fluorine, vanadium, and other elements of potential value) of the "inferred" class over the entire region and, in certain areas, reserves of the "indicated" class as well. These data should thus provide sufficient basis for industry to make preliminary selection of sites and of combinations of beds suitable for mining, but the detailed sampling necessary to prove (or measure) reserves and the drilling and large-scale mapping (i.e. 50 to 100 ft to the inch) necessary to work out details of structure in the degree necessary to plan actual mining operations will have to be done by the companies themselves. We feel this is the practical as well as the ideal division of effort between federal and private geological studies, for though the federal investigations outline the regional picture in a detail which facilitates and stimulates the work of private industry, it leaves to industry those studies incident to actual production.

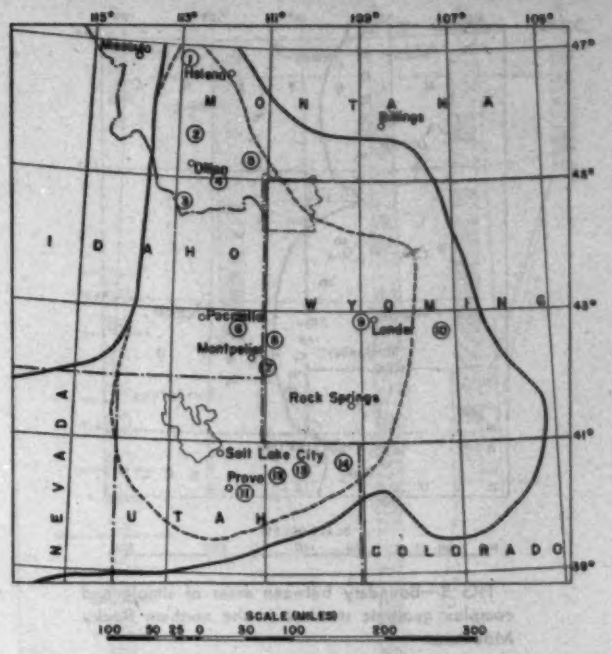

FIG 1—Index map of the western phosphate field, showing limits of the Phosphoria, Park City, and Ember formations (solid line) and their phosphate deposits (dashed line).

The eastern and southeastern boundaries shown on the map fairly accurately represent the true limits at the time of deposition. The western limits at the time of deposition cannot now be accurately reconstructed because the Permian rocks there either have been eroded away or are concealed beneath thrust plates of older rocks. The southern and southwestern limits are now poorly defined because not enough stratigraphic work has been done to differentiate the facies of the Permian Phosphoria formation from those of the Kaibab limestone, Gerster formation, and Arcturus formation and other western facies which may be at least partial age equivalents. Numbers show the location of measured sections on Fig 3, 4 and 5.

Though it is not possible to give in this paper a full account of even new information acquired on the geology of the western field, some of the more salient features of the regional geology and the phosphate deposits are discussed in the following pages. In addition, the occurrence of one of the other trace constituents—uranium—is discussed briefly in another paper.[1]

Geologic Setting of the Western Phosphate Field

Phosphate deposits in the Permian Phosphoria formation and its partial stratigraphic equivalent in Utah, the Park City formation, crop out in an area of some 100,000 square miles in Montana, Idaho, Wyoming, Utah, and Nevada* (Fig 1). This area may be divided into an eastern portion of relatively simple geologic structure and a western portion of complex geologic structure. The boundary between the two provinces is irregular but roughly approximates meridian 111° (Fig 2).*

In the eastern part of the field the dips are gentle and the folds are the open-dome and basin type which have no dominant orientation. Thus the Uinta Mountains of Utah and the Centennial Range of Montana strike east, the Wind River Mountains and the Gros Ventre Range of Wyoming strike northwest, and the Rock Springs anticline of Wyoming strikes north. Though numerous faults, including some thrusts, are found, large unfaulted areas are present also. Eastward, near and beyond the margin of the Phosphoria formation, the folds become less numerous, even less complex, and gradually die out toward the plains.

[1] References are at the end of this paper.

* Discovered recently in the Goshute Range by H. E. Wheeler, personal communication.

* Details of the regional structure can be seen on the many published quadrangle maps over the field; the general picture, however, is best shown on the Tectonic Map of the United States: Amer. Assn. Petr. Geol., 1944, and the Geologic Map of the United States: U. S. Geol. Survey, 1932.

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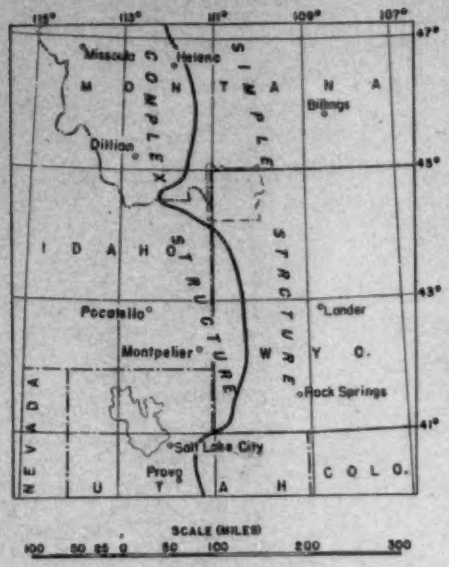

FIG 2—Boundary between areas of simple and complex geologic structure in the northern Rocky Mountains.

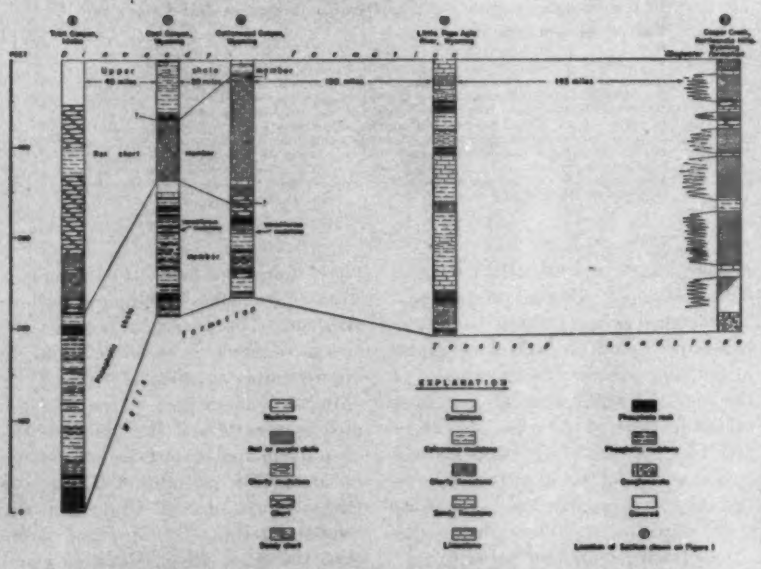

FIG 3—Typical sections of the Phosphoria formation in Wyoming and Idaho.

The Rattlesnake Hills section was measured by H. D. Thomas;[4] the Lander section by Ralph H. King;[11] the Salt River Range section by J. D. Love and L. E. Smith; the Coal Canyon section by V. E. McKelvey, and the Trail Canyon section by L. E. Smith, R. A. Hoppin and V. E. McKelvey, all of the U. S. Geological Survey.

In marked contrast, the western portion of the field is characterized by steep dips and tight, closely spaced folds, many of which are overturned. Overthrust, reverse, and transverse faults of both large and small displacement are abundant—in fact it is unusual to find outcrop segments of more than a mile in length which are not broken by faults, and it is not uncommon to find outcrops of the Phosphoria formation so faulted and mashed as to be altogether unsuitable for mining. Normal or valley faults of large displacement and relatively recent origin are also abundant in the western portion of the field, and, though it is not generally so classed in textbooks, this part of the region is in reality a part of the Basin and Range structural province.

The structure of the western part differs somewhat from east to west across the field. Near the eastern margin, as in the Wasatch Mountains of Utah, the Salt River Range of Wyoming, and the Tendoy Mountains of Montana, most of the folds are overturned, many of them isoclinal and of very short wavelength; the mountains or the folded belts are tens of miles in width and are identifiable for scores of miles along the strike, whereas the structural valleys are narrow or of the order of only a few miles in maximum width. In addition, the rocks cropping out at the surface are predominantly of late Paleozoic and Mesozoic age. Farther west, on the other hand, the folds are more open, fewer of them are overturned and their wavelength is longer; the folded belts or mountain ranges are less continuous along the strike and in much of the region (south-central Idaho, western Utah, and eastern Nevada) the basins are wider and occupy a larger proportion of the region than to the east. Although late Paleozoic and Mesozoic rocks are found, the proportion of pre-Carboniferous rocks cropping out at the surface is much higher than to the east. The structure is complex, however, because of the abundance of large overthrust faults, and, in fact, it is these overthrusts that are in large part responsible for the numerous outcrops of the pre-Carboniferous rocks.

Regional Variations in the Phosphoria Formation

The Phosphoria formation is noted for the great lateral continuity of the beds composing it, but it displays significant lithologic variations over the field as a whole. At its type locality in southeastern Idaho,[3] the Phosphoria formation consists of a lower, phosphatic shale member about 180 ft thick and an upper member, the Rex chert member, 240 ft thick; another member, a cherty mudstone or shale 15 to 75 ft thick overlies the Rex member in most of southeastern Idaho, though it is not well-defined at the type locality of the Rex member in the Crawford Mountains of Utah.[3] These units of the Phosphoria are easily recognizable over a wide area in Idaho and adjoining parts of Wyoming and Utah, but in central Wyoming the whole aspect of the formation is different, for it is thinner, and contains a greater pro-

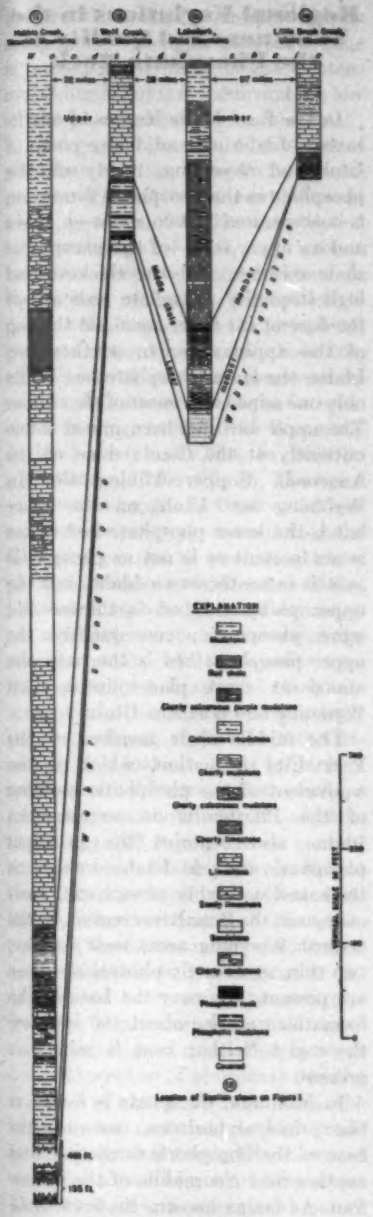

FIG 4—Typical sections of the Park City formation in Utah.

The Little Brush Creek section was measured by D. M. Kinney; the Lake Fork, and Wolf Creek sections by J. W. Huddle; and the Hobble Creek section by A. A. Baker, R. S. Sears, M. D. Stewart, G. F. Hosford, and D. P. Sprouse, all of the U. S. Geological Survey.

Note: the line at the top of the Park City formation may not be the same time-line at every locality, for the upper part of the Park City formation tongues out into red beds similar to the Woodside shales in the eastern part of the area; see H. D. Thomas and M. L. Krueger.[8]

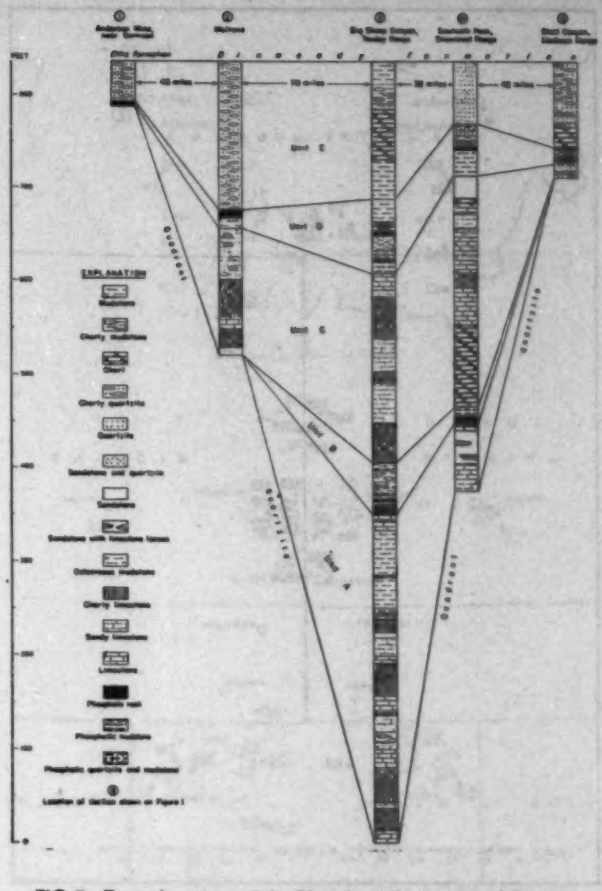

FIG 5—Typical sections of the Phosphoria formation in Montana.

[1]The Madison Range section was measured by R. A. Swanson; Sawtooth Peak section by F. S. Honkala and O. A. Payne; Dell section by W. R. Lowell; and the Melrose and Garrison sections by M. R. Klepper, all of the U. S. Geological Survey. Stratigraphic correlation by M. R. Klepper.

portion of sand and carbonate and much less phosphate and shale. Farther east, in southeastern Wyoming, the phosphate is entirely absent and the formation tongues out into nonmarine red beds[4] (Fig 3). Although the better-known phosphate deposits lie east of Fort Hall in Idaho, the Phosphoria formation has been identified farther west near Malta, Idaho.[6] Both the lower phosphatic shale member and the Rex chert member are reported to be present there and to total about 900 ft in thickness.[6] The quality and thickness of the phosphate beds, however, are unknown.

The westward thickening of the Phosphoria formation is displayed in the other states and so, to some extent, are the other lithologic variations. In Utah the stratigraphic unit known as the Park City formation is the partial equivalent of the Phosphoria, and at Park City, its type locality,[7] it is about 590 ft thick and consists of a lower limestone member, which may be stratigraphically equivalent to the upper part of the Wells formation in southeastern Idaho; a middle shale member (phosphatic, but containing no high-grade phosphate beds) equivalent to the phosphatic shale; and an upper limestone member, equivalent to the Rex chert member of the Phosphoria formation of southeastern Idaho. Eastward the lower member thins out, the phosphate deposits disappear, and the shale and upper limestone members are more clastic and finally tongue out into nonmarine red beds in eastern Utah and western Colorado[8,9] (Fig 4). As in Idaho, the Park City formation thickens markedly to the west and contains a greater thickness of chemical precipitates. A section recently measured by Newell[10] in the Confusion Range near the western border of Utah is 4500 ft thick, though its phosphate content is unknown. In southwestern Montana new investigations by M. R. Klepper, W. R. Lowell, A. P. Butler and other geologists of the Geological Survey show that the Phosphoria formation consists of from two to five lithologic units (Fig 5), provisionally termed

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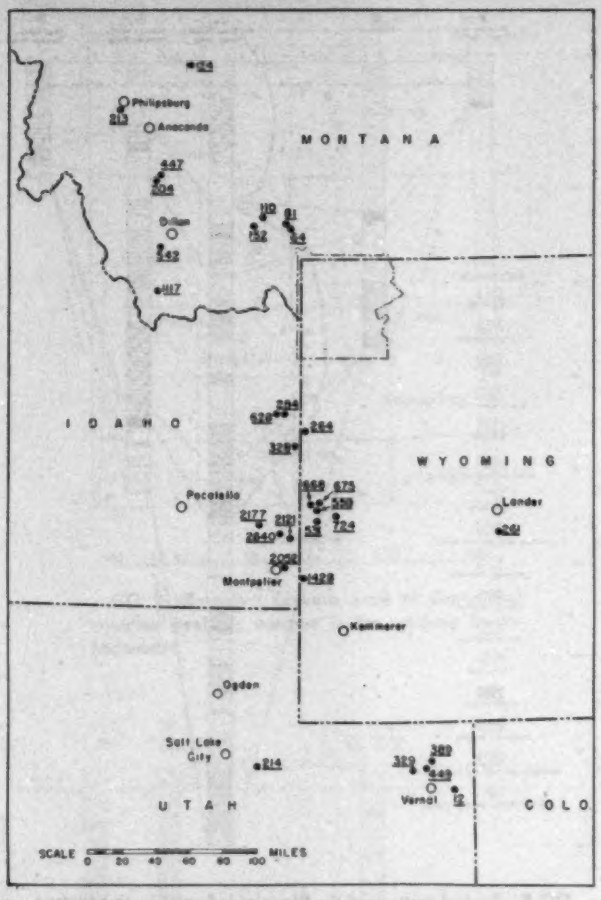

FIG 6—Total phosphate (in feet times percentage of P_2O_5) in the phosphatic portion of the Phosphoria and Park City formations.

units *A*, *B*, *C*, *D*, and *E*. Unit *A*, at the base, consists of a sequence of cherty carbonate and clastic rocks which may prove to be the equivalent of the upper part of the Wells formation in southeastern Idaho, and the lower limestone member of the Park City formation of Utah. Unit *B* is composed of phosphate rock and phosphatic mudstone, unit *C* consists mainly of carbonate rock, unit *D* of phosphatic mudstone, and unit *E* of chert. In the Centennial Range the upper chert, unit *E*, is overlain by a thin section of dark-colored mudstone, which has not been recognized elsewhere in Montana. In the vicinity of Dell and Lima the total thickness of the Phosphoria is 485 ft. The clastic-carbonate sequence (unit *A*), the lower phosphate (unit *B*), and the lower carbonate (unit *C*) disappear eastward in the Madison Range and northward in the Garrison-Drummond area.

In summary, the Phosphoria formation along the eastern margin of the marine basin in which it was deposited contains no phosphatic rocks and consists of thin marine layers, principally carbonate rock and sandstone, interbedded with nonmarine red beds. Westward, as in central Wyoming and eastern Utah, thin phosphatic rocks are interbedded with limestone, mudstone, and sandstone; still farther west, as in western Utah and southeastern Idaho, the formation thickens, and the bulk of it is composed of chemical precipitates (limestone and phosphate rock), very fine detritus (clay and silt), and organic matter. In Idaho the thickening of the formation is in a large part a result of the increased thickening of the phosphatic rocks; in Utah and Montana it is due largely to increase in the amount of fine detritus and carbonate and, though the amount of phosphate is greater too, it is diluted by the other sediments; the phosphatic rocks are not as high in quality, therefore, as they are in Idaho.

Regional Variations in the Thickness and Quality of the Phosphate Rock

In the Bear River Region of southeastern Idaho and adjoining parts of Utah and Wyoming, nearly all the phosphate in the Phosphoria formation is concentrated in two zones—a lower and an upper zone—of the phosphatic shale member, and the thickest and highest-quality phosphate beds are at the base of the lower zone and the top of the upper zone. In southeastern Idaho, the lower phosphate bed is the only one mined over most of the region. The upper bed has been mined intermittently at the Conda mine of the Anaconda Copper Mining Co. In Wyoming and Utah, on the other hand, the lower phosphate bed either is not present or is not as phosphatic as it is in southeastern Idaho, but the upper phosphate bed is thicker and more phosphatic; consequently the upper phosphate bed is the only one mined at most places in western Wyoming and northern Utah.

The middle shale member of the Park City formation, which is the equivalent of the phosphatic member of the Phosphoria in southeastern Idaho, also contains the principal phosphatic beds in Utah; none is as thick and as highly phosphatic, however, as in the Bear River region. In the central Wyoming area, near Lander, two thin, moderately phosphatic zones are present, one near the base of the formation and one about 100 ft below the top.[11] Neither zone is mined at present.

In Montana, phosphate is found at two principal horizons, one near the base of the Phosphoria formation, and another near the middle of the formation. As far as known, the lower zone is of commercial interest only in the Snowcrest and Centennial Ranges, where it is 4 to 6 ft thick, and contains about 32 pct P_2O_5. Elsewhere it is too thin (as in the vicinity of Dillon), or too low in phosphate content (as in the vicinity of Dell) to be mined at present, or is absent altogether, as in the Madison Range, and the Melrose, Phillipsburg, and Garrison areas. The upper phosphate bed is about 4 ft thick and contains about 32 pct P_2O_5 in the Garrison area (where, because the lower members are absent, it occurs at the base of the formation), but farther south in the Melrose, Dillon, and Dell areas the phosphate is so diluted by interbedded and admixed mudstone,

that no thick, highly phosphatic beds are present. The upper phosphate zone is present in the Madison and Centennial Ranges, but is too thin and too low in phosphate content to be mined.

The general westward increase of phosphate content is shown by a comparison of the total amount of phosphate, in terms of thickness multiplied by percentage of P_2O_5, at various localities over the region. Thus, in the vicinity of Lander, the total feet-percent of phosphate in the phosphatic portions of the formation is about 260, and the total increases progressively westward to about 600 ft-pct in western Wyoming, 1400 on the Wyoming-Idaho border, and 2000 to 2600 in southeastern Idaho (Fig 6). Though a similar trend exists in Utah and Montana, at no place yet known does the formation contain as much total phosphate as it does in southeastern Idaho.

Total phosphate alone, of course, is not a reliable means of evaluating the relative quality of the deposits over the region, for a large total phosphate content can be dispersed over a large thickness of rock too low in P_2O_5 content to be minable, as it is in parts of southwestern Montana. Conversely, a relatively small total amount of phosphate, say 160 ft-pct, can be concentrated in one minable layer, as it is in the Garrison area and Centennial Range, where virtually all the phosphate in the formation is concentrated in one minable bed belonging to the middle and lower zones, respectively. Over the remainder of the field, however, there is good correlation between the total amount of phosphate and the thickness of high-grade beds. Thus, the Lander area contains no high-grade beds at all, and the grade increases westward to southeastern Idaho where 11 to 19 ft of beds contain 31 pct or more P_2O_5 (Fig 7). So it is too with beds of lower phosphate content; in the Lander area about 3 ft of beds contain over 25 pct P_2O_5 and in southeastern Idaho 20 to 32 ft of beds contain more than 25 pct P_2O_5 (Fig 8). Similarly in the Lander area only about 4 ft of beds contain more than 18 pct P_2O_5, whereas in southeastern Idaho 43 to 62 ft of the section contains more than that amount (Fig 9).

The conclusion that the Bear River region contains the most and best phosphate is inescapable, but the fact that significant production is coming from Montana now, and that large phosphate reserves exist there and in other parts of the field should not be overlooked. One high-quality phosphate bed only 4 ft thick in a favorable structural and geographic situation can be mined, even though there may be no other phosphatic beds in the section. Regional trends in phosphate content are therefore not to be relied on too heavily in selecting potential mine sites. Nothing short of an exposed and sampled section will prove conclusively that a minable bed is not present at any given locality over a large part of the field.

Though regional variations in the stratigraphy must be considered in an appraisal of the potentialities of the field, they should not be emphasized so as to overshadow one of the most important features of the Phosphoria formation, namely, the remarkable lateral continuity of individual layers in particular areas. Thus, a distinctive phosphate bed at the contact between the Phosphoria formation and the underlying Wells formation in the Bear River region is identifiable from the Crawford Mountains of Utah to Fort Hall, Idaho, a distance of more than 100 miles, although the bed is never over a few inches thick. The lower phosphate bed is mined at various localities between Montpelier and Fort Hall, and over this distance it is relatively uniform in thickness and lithology. Many similar examples of great lateral continuity of individual layers could be given. The lateral continuity of the beds is not great in the eastern and northern parts of the field, but even there the lateral continuity in most places is sufficient to greatly simplify evaluation of grade and tonnage of any particular layer. In contrast with most other types of mineral deposits, as few as 8 or 10 carefully cut samples are generally adequate to evaluate the phosphate content of any particular bed along an outcrop as much as a mile or two in length.

Early reserve estimates[12-15] were based upon reconnaissance data and calculated in such a way as to show

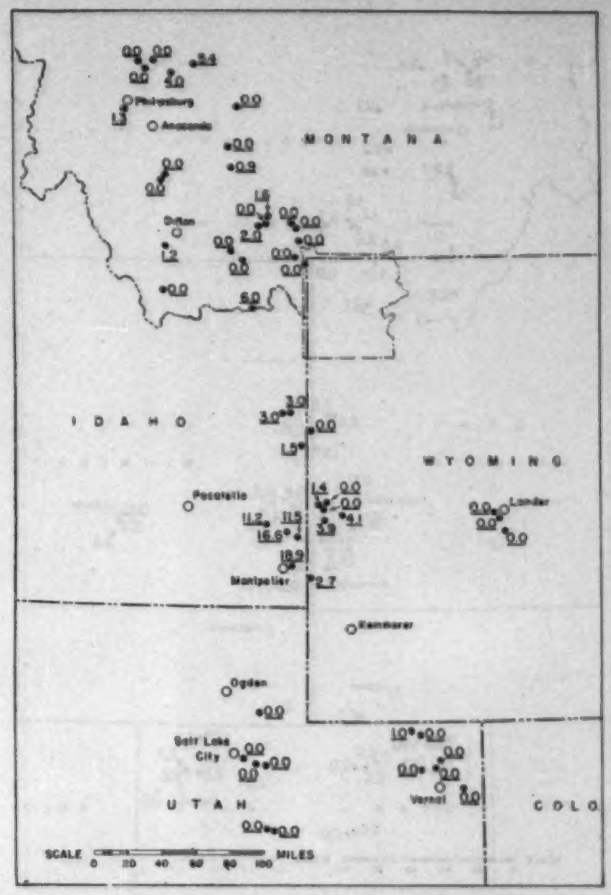

FIG 7—Total thickness (in feet) of rocks in the Phosphoria and Park City formations containing more than 31 pct P_2O_5.

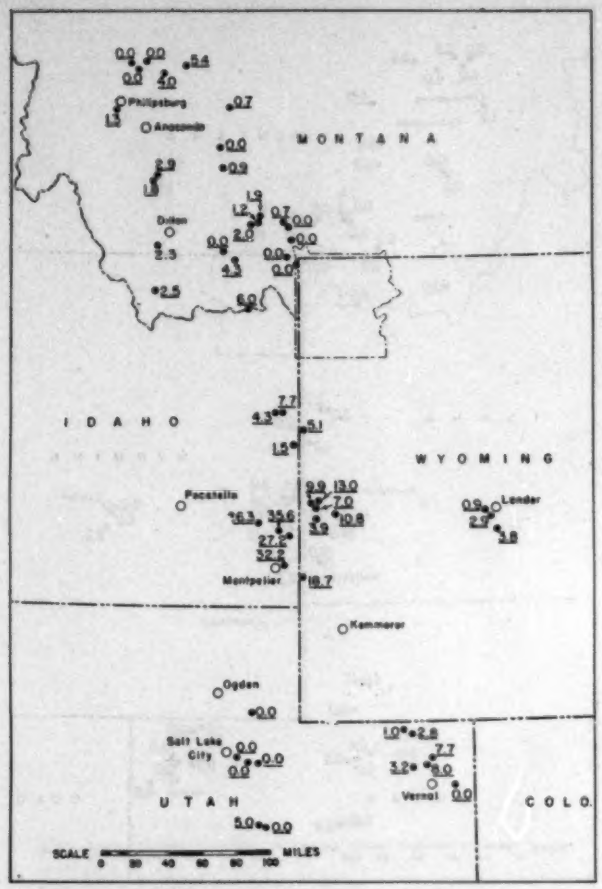

FIG 8—Total thickness (in feet) of rocks in the Phosphoria and Park City formations containing more than 25 pct P_2O_5.

the amount of phosphate that might be available ultimately. For the most part only the high-quality rock was included in the estimates; in keeping with the quality of the data upon which the estimates were based, no refinements were introduced in the methods of calculation. These estimates served a very useful purpose in supplying legislators and mineral economists with data upon which to form policy on phosphate export, the administration of public lands, and other matters of national importance. But the early estimates are of little value in judging the potentialities of the field in terms of present mining practices and costs or in selecting deposits for mining.

Since mining activity has been expanded in the western field the Geological Survey has begun to acquire data which will permit estimation of reserves on a more realistic basis. Such estimates will not be completed for many years. Table 1, showing preliminary estimates of reserves of rock minable under present conditions, is presented with some reluctance, for—and it cannot be overemphasized—the estimates and data are incomplete.

The data available show that reserves of high-grade rock in deposits suitable for low-cost mining are limited indeed, known deposits contain a little more than 25 million tons, and it is improbable that new discoveries and appraisals will much more than double the figure. Reserves of high-grade rock suitable for underground mining (above entry level) are likewise surprisingly limited, 350 million tons in known and appraised deposits. Large tonnages, perhaps equaling or exceeding that of the Bear River region, of high-grade rock in beds 3 to 4 ft thick are present in the mountainous area of western Wyoming. But, making due allowance for rock in as yet unappraised deposits, it is highly unlikely that the western field as a whole contains more than a billion tons of rock containing more than 31 pct P_2O_5 in deposits minable by present practices. In fact, if the cost of transportation and underground mining persists at the present level in the future, it is probable that more favorably situated lower-grade deposits will be mined far in advance of the largest part of the potential billion tons of high-grade rock present over the field.

A much larger tonnage of rock containing 24 to 31 pct P_2O_5 is known. For example, 75 million tons in deposits suitable for stripping and a billion tons

Table 1 . . . Preliminary and Incomplete Estimate of Reserves of Phosphate Rock Suitable for Mining in Permian Rocks of the Western Field[a]

(Millions of Short Tons)

Area	Suitable for Open-cut Mining		Suitable Only for Underground Mining (Above Entry Level)	
	+31 Pct P_2O_5	24–31 Pct P_2O_5	+31 Pct P_2O_5	24–31 Pct P_2O_5
Montana	Not estimated[b]		50[c]	75[d]
Teton Basin, Wyoming and Idaho	None	Not estimated	Not estimated[e]	
Western Wyoming	Not estimated		Not estimated[f]	
Bear River region, Idaho, Wyoming and Utah	25	75	300	1,000
Lander area, Wyoming	None	Not estimated	None	25[g]
Central Wyoming, excluding Lander area	None	Not estimated	None	Not estimated[h]
Utah, excluding Bear River region	None	None[i]	None	Not estimated[j]

[a] All estimates based upon meager data, both as regards topographic and geologic setting of individual deposits as well as chemical analyses. The estimates are of rock in the ground and no allowance is made for loss due to dilution or unfavorable structural conditions beyond those which can be judged from surface outcrops. All figures are rounded off to the nearest 25 million tons. Minimum mining width assumed to be 3 ft in both open-cut and underground mining.

[b] The only potential open-pit deposits known in Montana are in the Centennial Range, and in the area southwest of Lima. Though these deposits appear promising, not enough data are available to determine whether or not they can be mined by open-pit methods, or what their reserves may be.

[c] Estimated by M. R. Klepper; includes deposits in the Centennial Range, part of which may be amenable to open-pit mining.

[d] Estimated by M. R. Klepper.

[e] Minable rock containing more than 31 pct P_2O_5 is present in the Big Hole Mountains, where there may be a few million tons; rock containing 24 to 31 pct P_2O_5 may total several million tons.

[f] Reserves in both classes probably total scores of millions of tons.

[g] Modified from estimates made by R. H. King.[11]

[h] May total tens of millions of tons.

[i] A few small deposits near Provo are estimated to contain less than 1 million tons.

[j] May total a few million tons.

suitable for underground mining are estimated in the Bear River region alone; this quantity may well be quadrupled when deposits in the remainder of the field are appraised.

The tonnage of rock containing more than 18 pct P_2O_5, though not yet estimated in even preliminary fashion, is vast indeed, and may be measurable in billions, perhaps even tens of billions of tons. For example, the southeast flank of the Uinta Mountains, near Vernal, contains several hundred million tons of rock with slightly more than 20 pct P_2O_5, most of which can be mined by open-cut methods. In the Bear River region the whole phosphatic shale member of the Phosphoria formation averages nearly 12 pct P_2O_5; and a thickness of about 70 ft, in the lower and upper zones together, averages 20 pct P_2O_5 (Fig 10). If these units were mined and beneficiated, nearly two-thirds of the phosphate in the formation would be recovered and a tonnage of perhaps 3 billion tons would be available.

Additional large reserves of both high- and low-grade rock would be available below entry level if shaft mining were to become practicable. In short, potential reserves of phosphate in the region are vast, and probable advances in mining and metallurgical technology may be expected to expand our concept of what constitutes minable reserves. Thus, at the present rate of production it would be only a few years until reserves in open-cut mines in the Bear River region were exhausted; but every operating company has research in progress which promises to enable them to mine lower-grade rock on their own properties before the deposits now being mined are exhausted.

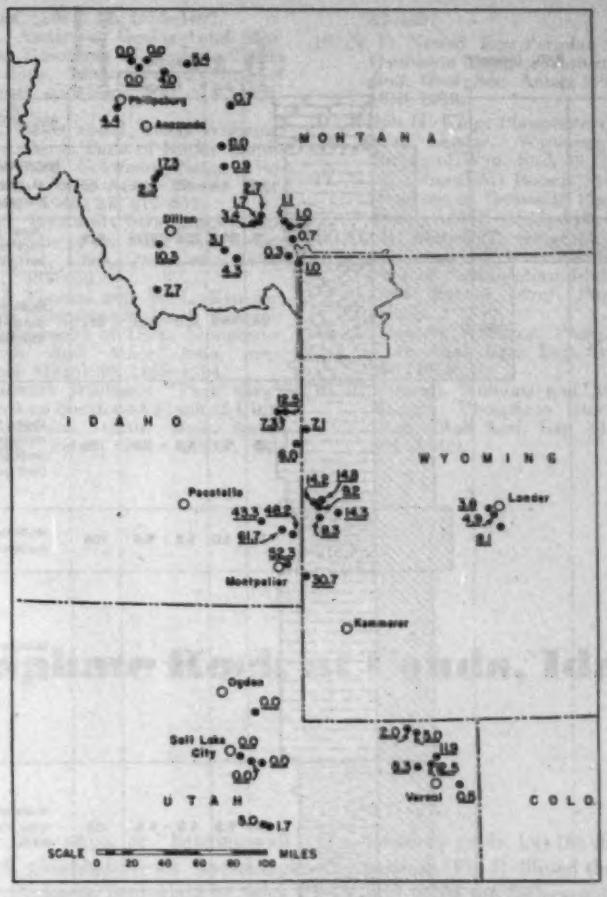

FIG 9—Total thickness (in feet) of rocks in the Phosphoria and Park City formations, containing more than 18 pct P_2O_5.

Conclusions

The western phosphate field may be divided into an eastern and western part. In the eastern part the geologic structure is simple, the folds are open and widely spaced, and the beds gently dipping; the phosphate deposits are for the most part thin and of relatively low quality. As a result of this structural pattern, outcrops of the Phosphoria formation are widely spaced, but are long and continuous; the cross-sectional area (along the dip of beds) above entry level is very large in individual areas and reserves of phosphatic rock are accordingly large, but of low quality; relatively large tonnages of rock amenable to open-pit mining are known and additional large tonnages will be discovered; underground mining over much of the area will be costly because of the thinness of the beds, although such increased costs may be somewhat offset by the lack of structural complexitites. The future of mining in the eastern part of the field lies in the discovery of areas where one or another of the phosphate beds is above average thickness and quality, as in the Centennial Range, and in the discovery and development of open-pit deposits where the cost of mining is low enough to offset the cost of beneficiating the lower grade rock.

In the western part of the field the geologic structure is complex; the folds are tight, closely spaced, and many of them overturned; the beds dip steeply, and faults of large and small displacement are abundant; the phosphate deposits, however, are thick and of high quality over much of the area, especially in the Bear River region. Outcrops of the Phosphoria formation are therefore closely spaced, but are short and discontinuous; though most individual deposits are small, total reserves are large because of repetition of the beds and the relatively great thickness of the phosphatic beds; only relatively small tonnages of high-quality rock minable by open-cut methods are available and few large additional deposits will be discovered. Underground mining of choice deposits may be expected to be as cheap as anywhere in the region, but will be expensive in many areas because of structural complexities. Additional valuable phosphate deposits may be discovered to the west in central Idaho, in the southwestern corner of Montana, in western Utah, and eastern Nevada; if the trend of westward thickening of the phosphate beds continues to those areas.

Our present understanding of the geology of the western field indicates that reserves of phosphate minable under existing economic conditions are

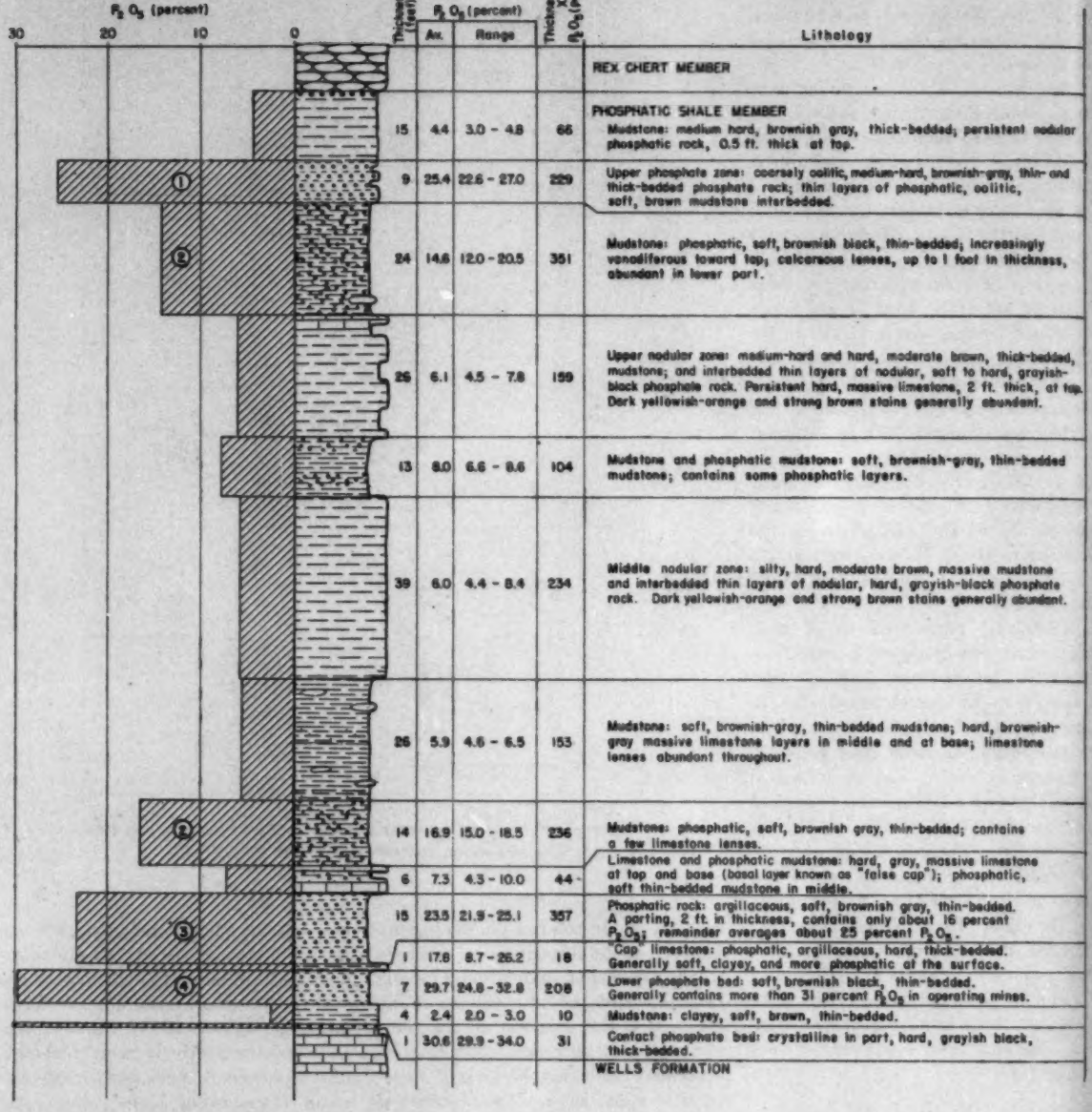
P_2O_5 (percent)
30
20
10
0
Thickness (feet)
P_2O_5 (percent)
Av.
Range
Thickness (ft) X (P_2O_5 (percent))
Lithology
REX CHERT MEMBER
PHOSPHATIC SHALE MEMBER
15 4.4 3.0 - 4.8 66
Mudstone: medium hard, brownish gray, thick-bedded; persistent nodular phosphatic rock, 0.5 ft. thick at top.
9 25.4 22.6 - 27.0 229
Upper phosphate zone: coarsely oolitic, medium-hard, brownish-gray, thin- and thick-bedded phosphate rock; thin layers of phosphatic, oolitic, soft, brown mudstone interbedded.
24 14.6 12.0 - 20.5 351
Mudstone: phosphatic, soft, brownish black, thin-bedded; increasingly vanadiferous toward top; calcareous lenses, up to 1 foot in thickness, abundant in lower part.
26 6.1 4.5 - 7.8 159
Upper nodular zone: medium-hard and hard, moderate brown, thick-bedded, mudstone; and interbedded thin layers of nodular, soft to hard, grayish-black phosphate rock. Persistent hard, massive limestone, 2 ft. thick, at top. Dark yellowish-orange and strong brown stains generally abundant.
13 8.0 6.6 - 8.6 104
Mudstone and phosphatic mudstone: soft, brownish-gray, thin-bedded mudstone; contains some phosphatic layers.
39 6.0 4.4 - 8.4 234
Middle nodular zone: silty, hard, moderate brown, massive mudstone and interbedded thin layers of nodular, hard, grayish-black phosphate rock. Dark yellowish-orange and strong brown stains generally abundant.
26 5.9 4.6 - 6.5 153
Mudstone: soft, brownish-gray, thin-bedded mudstone; hard, brownish-gray massive limestone layers in middle and at base; limestone lenses abundant throughout.
14 16.9 15.0 - 18.5 236
Mudstone: phosphatic, soft, brownish gray, thin-bedded; contains a few limestone lenses.
6 7.3 4.3 - 10.0 44
Limestone and phosphatic mudstone: hard, gray, massive limestone at top and base (basal layer known as "false cap"); phosphatic, soft thin-bedded mudstone in middle.
15 23.5 21.9 - 25.1 357
Phosphatic rock: argillaceous, soft, brownish gray, thin-bedded. A parting, 2 ft. in thickness, contains only about 16 percent P_2O_5; remainder averages about 25 percent P_2O_5.
1 17.8 8.7 - 26.2 18
"Cap" limestone: phosphatic, argillaceous, hard, thick-bedded. Generally soft, clayey, and more phosphatic at the surface.
7 29.7 24.8 - 32.8 208
Lower phosphate bed: soft, brownish black, thin-bedded. Generally contains more than 31 percent P_2O_5 in operating mines.
4 2.4 2.0 - 3.0 10
Mudstone: clayey, soft, brown, thin-bedded.
1 30.6 29.9 - 34.0 31
Contact phosphate bed: crystalline in part, hard, grayish black, thick-bedded.
WELLS FORMATION
①
②
③
④

FIG 10—Generalized section of phosphatic shale member of Phosphoria formation in the Slug Creek Quadrangle, Idaho.

The phosphate content is based upon analyses of samples collected from a crosscut on the 300 level of the Conda mine; and trenches in Trail Canyon and south Dry Valley. Unit 1 contains 10.7 pct of total phosphate in the phosphatic shale member; unit 2 contains 26.6 pct of the total; unit 3, 16.2 pct; and unit 4, the only bed generally mined at present, 9.1 pct.

large, but by no means as large as earlier estimates would indicate. As a source of phosphate to be mined in the distant future from lower-grade beds and from shaft mines, however, the Phosphoria formation contains an even larger supply than indicated by earlier estimates.

Acknowledgments

Abundant thanks are due members of the Anaconda Copper Mining Co., the San Francisco Chemical Co., the Simplot Fertilizer Co., the Montana Phosphate Products Co., the Humphreys Phosphate Co., the American Smelting and Refining Co., and many residents of the field who gave access to property, furnished information and otherwise generously cooperated in our western phosphate investigations. I am grateful also to many of my associates on the Geological Survey, particularly W. W. Rubey, for stimulating advice and criticism extended over a period of several years; special thanks are due M. R. Klepper, J. Steele Williams, and A. A. Baker, who furnished part of the information included in this paper, and to R. A. Harris, R. A. Smart, Gordon Waring, and Earl Sparks, who helped compile some of the data shown on the illustrations.

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Mining of Phosphate Rock at Conda, Idaho

By T. C. RUSSELL,* Member AIME

The Conda phosphate mine, eight miles north of Soda Springs in Caribou County, Idaho, was opened up by the Anaconda Copper Mining Co. in 1920. Except for brief periods, during the 20's and early 30's, the mine has maintained a continuous production, and 2,100,000 short tons of phosphate rock have been shipped to the company's phosphate fertilizer manufacturing plant at Anaconda, Mont., or to independent processing plants on the Pacific Coast; a small tonnage of the rock has been shipped to the Hawaiian Islands.

The Phosphoria formation on the Conda property consists of two, commercial-grade phosphate beds separated by 147 ft of black to brownish colored phosphatic calcareous shales interbedded with softer mudstones, clays, and occasional narrow seams of oölitic phosphate. The main lower phosphate bed in the series averaging 7.1 ft in thickness, is called the "footwall bed," which lies conformably on the Wells limestone formation, the latter often containing irregular bands of chert. The upper or "hanging-wall bed" of phosphate rock averaging 7.2 ft in thickness, is overlaid by thick quartzite beds, locally called the "Rex chert" member in the upper portion of the Phosphoria formation. (Fig 1.)

There are two anticlines on the property with axis striking approximately parallel to the mountain range beneath which extensive development on phosphate rock has been conducted in the past, and is now in progress. The phosphate beds have been eroded to a large extent from the tops of these anticlines, leaving at least three well-pronounced outcrops of the phosphate beds exposed within the boundaries of the Anaconda Copper Mining Company's properties. At present, mining operations in the Conda No. 3 mine are confined to the two phosphate beds along the east flank of the easternmost anticline, where the dips on the beds vary from 50° to 60° east, thereby affording favorable conditions for the sublevel mining system now in use. Mining operations formerly were confined to the footwall or lower oölitic phosphate bed because of its higher phosphate and vanadium contents (32 pct P_2O_5 and 0.29 pct V_2O_5); this bed averages 7.1 ft in thickness. The oölitic portions of the upper bed were of satisfactory grade, but the shale and clay partings (Fig 2) diluted the mined rock and could not be separated economically by selective mining or by sorting. Experiments in beneficiating this rock were carried on for a number of years. Recently, a simple washing process was adopted, which gave a satisfactory grade of concentrates and a high recovery of P_2O_5.

In 1947 it was decided to expand Anaconda's production of treble superphosphate and phosphoric acid by approximately 90 pct. This rock will be obtained from the footwall and hanging-wall beds, and the rock from the hanging-wall bed will be beneficiated at Anaconda.[1]

In order to take advantage of the previous development on the 500 secondary haulage level, short crosscuts will be driven at approximately 950 ft intervals from the existing lateral drifts through the hanging-wall shale beds. From the inner end of each crosscut, short lateral drifts will be driven in both directions, so that four working raises can be put up on the dip of the phosphate bed at proper intervals along each of the lateral drifts. The mining operation on the footwall bed at Conda

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* Superintendent, Fertilizer Department, Anaconda Copper Mining Co., Conda, Idaho.

[1] References are at the end of this paper.

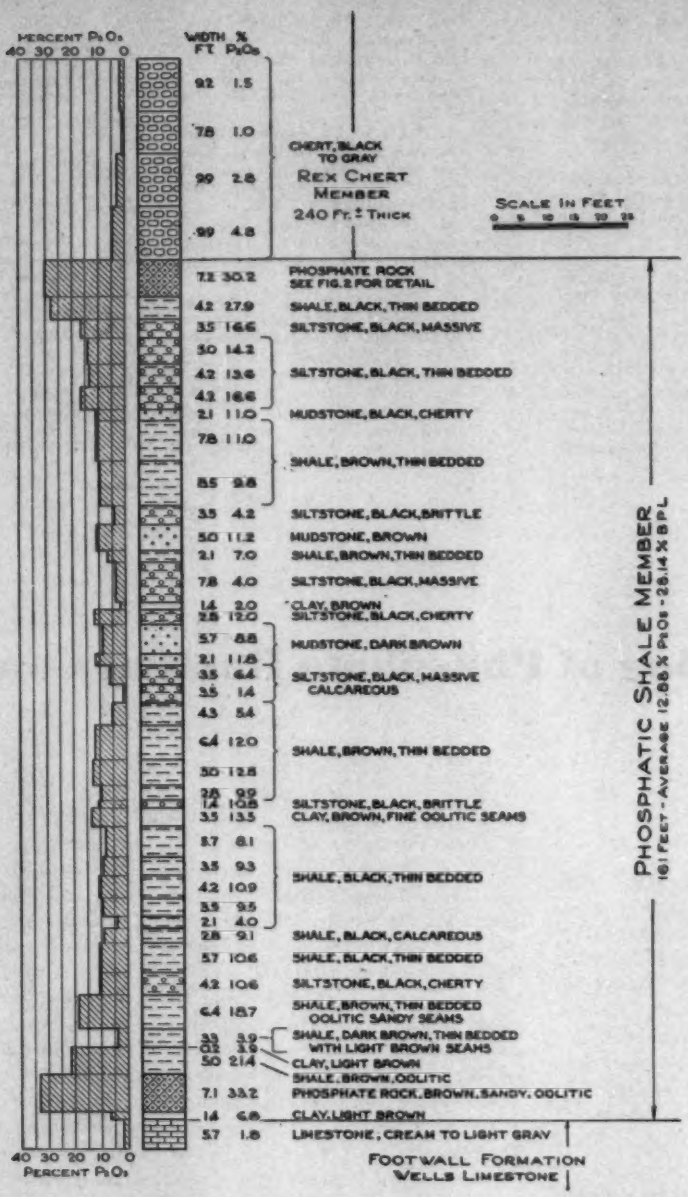
PERCENT P2O5
40 30 20 10 0
WIDTH FT
% P2O5
9.2 1.5
7.8 1.0
CHERT, BLACK TO GRAY
REX CHERT MEMBER
240 FT. ± THICK
9.9 2.8
9.9 4.8
SCALE IN FEET
0 5 10 15 20 25
7.2 30.2 PHOSPHATE ROCK SEE FIG. 2 FOR DETAIL
4.2 27.9 SHALE, BLACK, THIN BEDDED
3.5 16.6 SILTSTONE, BLACK, MASSIVE
3.0 14.2
4.2 13.6 SILTSTONE, BLACK, THIN BEDDED
4.2 16.6
2.1 11.0 MUDSTONE, BLACK, CHERTY
7.8 11.0
SHALE, BROWN, THIN BEDDED
8.5 9.8
3.5 4.2 SILTSTONE, BLACK, BRITTLE
5.0 11.2 MUDSTONE, BROWN
2.1 7.0 SHALE, BROWN, THIN BEDDED
7.8 4.0 SILTSTONE, BLACK, MASSIVE
1.4 2.0 CLAY, BROWN
2.8 12.0 SILTSTONE, BLACK, CHERTY
5.7 8.8 MUDSTONE, DARK BROWN
2.1 11.8
3.5 6.4 SILTSTONE, BLACK, MASSIVE CALCAREOUS
3.5 1.4
4.3 5.4
6.4 12.0 SHALE, BROWN, THIN BEDDED
5.0 12.8
2.8 9.9
1.4 10.8 SILTSTONE, BLACK, BRITTLE
3.5 13.5 CLAY, BROWN, FINE OOLITIC SEAMS
8.7 8.1
3.5 9.3 SHALE, BLACK, THIN BEDDED
4.2 10.9
3.5 9.5
2.1 4.0
2.8 9.1 SHALE, BLACK, CALCAREOUS
5.7 10.6 SHALE, BLACK, THIN BEDDED
4.2 10.6 SILTSTONE, BLACK, CHERTY
6.4 18.7 SHALE, BROWN, THIN BEDDED OOLITIC SANDY SEAMS
3.3 3.9 SHALE, DARK BROWN, THIN BEDDED WITH LIGHT BROWN SEAMS
0.2 3.5
5.0 21.4 CLAY, LIGHT BROWN
SHALE, BROWN, OOLITIC
7.1 33.2 PHOSPHATE ROCK, BROWN, SANDY, OOLITIC
1.4 6.8 CLAY, LIGHT BROWN
5.7 1.8 LIMESTONE, CREAM TO LIGHT GRAY
PHOSPHATIC SHALE MEMBER
101 FEET - AVERAGE 12.86 % P2O5 - 26.14 X B.P.L.
FOOTWALL FORMATION
WELLS LIMESTONE
40 30 20 10 0
PERCENT P2O5

FIG 1—Columnar section of Phosphoria formation as exposed in 509 crosscut, No. 3 mine, Conda, Idaho, showing variable P_2O_5 content.

has been previously described.[2] This bed lies on the Wells limestone, which provides a firm footwall, but the phosphate rock is overlaid by thin bedded phosphatic shales, which disintegrate readily. The mining conditions in the hanging-wall bed are reversed; here the footwall is composed of soft shale and clay and the hanging-wall is formed by the hard massive Rex chert. These new conditions were satisfactorily met by using a sublevel mining system. Three compartment raises are driven on the phosphate bed spaced at 196 ft centers. The sublevels are driven from the raises at 30 ft intervals along the dip and extend for 90 ft in each direction, as drifts on the bed. Stoping operations are started by raising through the 21 ft pillar of ore from the farther ends of the sublevels. The rock is then stoped out in a succession of steps (Fig 3). The overhead gob is supported by the stull and plank flooring of the previous slice. The soft footwall is held in place by 2 by 10 in. pine lagging placed as footboards under the stulls, which are used for staging and for temporary support for the unmined portion of the slice.

The phosphate rock is drilled with pneumatic feed-leg jackhammers, equipped with downstroke rotation. Auger drill steel, 1¼ in. in diameter, is used. Bits are V-shaped and the cutting edges are faced with stellite. Holes are drilled 6 ft deep across the width of the bed and the broken ore is blasted down to the sublevel and

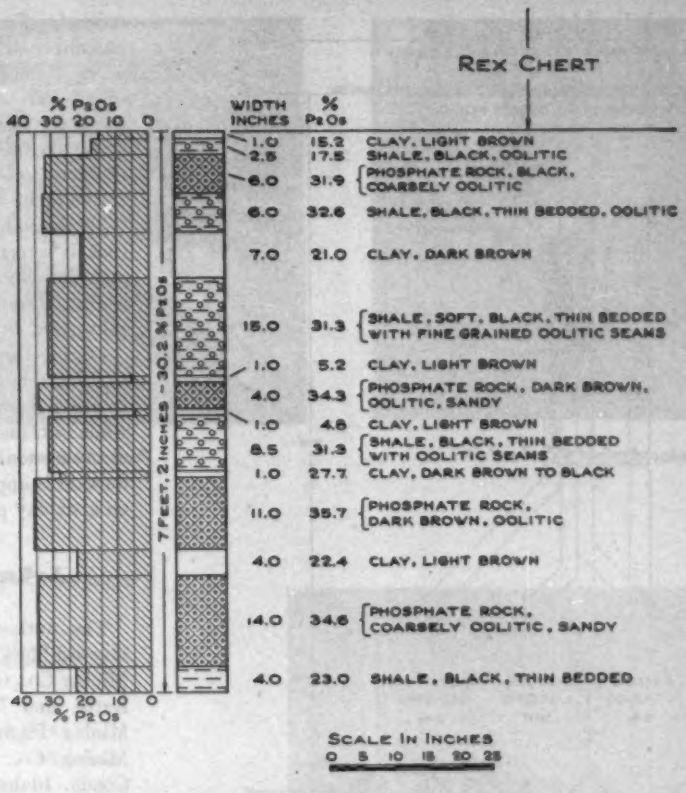
REX CHERT
% P2O5
40 30 20 10 0
WIDTH INCHES
% P2O5
1.0 15.2 CLAY, LIGHT BROWN
2.5 17.5 SHALE, BLACK, OOLITIC
6.0 31.9 PHOSPHATE ROCK, BLACK, COARSELY OOLITIC
6.0 32.6 SHALE, BLACK, THIN BEDDED, OOLITIC
7.0 21.0 CLAY, DARK BROWN
15.0 31.3 SHALE, SOFT, BLACK, THIN BEDDED WITH FINE GRAINED OOLITIC SEAMS
1.0 5.2 CLAY, LIGHT BROWN
4.0 34.3 PHOSPHATE ROCK, DARK BROWN, OOLITIC, SANDY
1.0 4.8 CLAY, LIGHT BROWN
8.5 31.3 SHALE, BLACK, THIN BEDDED WITH OOLITIC SEAMS
1.0 27.7 CLAY, DARK BROWN TO BLACK
11.0 35.7 PHOSPHATE ROCK, DARK BROWN, OOLITIC
4.0 22.4 CLAY, LIGHT BROWN
14.0 34.6 PHOSPHATE ROCK, COARSELY OOLITIC, SANDY
4.0 23.0 SHALE, BLACK, THIN BEDDED
7 FEET, 2 INCHES - 30.2 % P2O5
40 30 20 10 0
% P2O5
SCALE IN INCHES
0 5 10 15 20 25

FIG 2—Columnar section, detail of hanging-wall bed of the Phosphoria formation, as exposed in 509 crosscut of No. 3 mine, Conda, Idaho, showing P_2O_5 content.

slushed into the ore chutes on either side of the working raises. Anaconda-type scrapers of 10.5 cu ft capacity are used for slushing. A 15 hp, two speed, electric hoist is set up in the manway compartment of the raise to pull the scrapers from both directions. The slushing hoists are also used for hoisting timbers and other material. The rock is loaded into 2½-ton capacity side dump cars. The cars are hauled by 4-ton storage battery locomotives from the raise chutes to the station ore pockets in strings of 15 cars. All of the mine haulage tracks have a standard gauge of 3 ft. At the shaft station three cars can be dumped at each spotting of the train over the station ore pocket, from which the ore drops through an inclined pass into one of the vertical rock transfer compartments of the shaft (Fig 4). About 500 ft below the station the rock flows into one of the three cylindrical loading bins, which have been excavated above the main haulage level. Each of the bins has a capacity of 400 tons and is provided with four air-lift

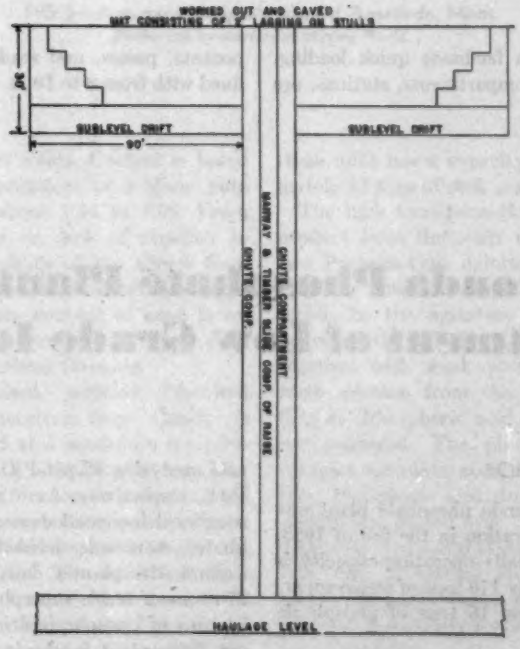

FIG 3—Long section of raise and sublevel stopes on hanging-wall bed at Conda, Idaho.

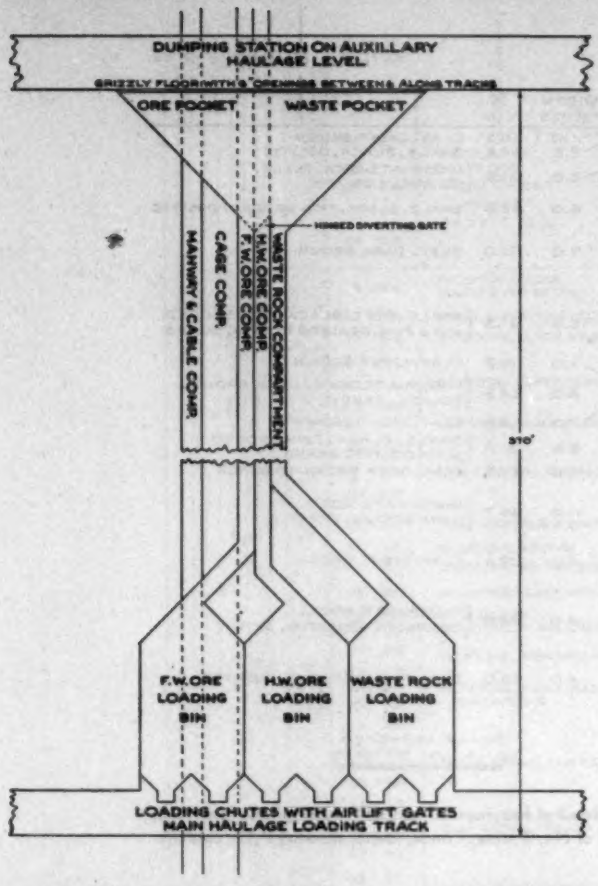

FIG 4—Elevation of shaft at Conda, showing arrangement for handling two classes of phosphate rock, and waste from auxiliary haulage to main haulage level.

gates, which facilitate quick loading. The shaft compartments, stations, ore pockets, passes, and loading bins are lined with from 8 to 10 in. of reinforced concrete. From the loading bins the hanging-wall rock, footwall rock, and waste is loaded into side dump cars of 11 tons capacity. These cars are trammed in ten car trips, by 20-ton storage battery locomotives, a distance of 8500 ft to the surface mill bins or to the waste dump.

At the mill the mine-run phosphate rock is screened over ¾ in. openings and the oversize is crushed in closed circuit through a swinging hammer pulverizer.

During the winter months the rock is dried from 6 to minus 2 pct moisture to prevent freezing in the railroad cars, while in transit to the phosphate plant at Anaconda. The present daily production is approximately 590 wet tons on a six day per week operation.

Acknowledgments

The author wishes to thank Mr. Edward Shea of the Anaconda Copper Mining Co., Geological Department at Butte, and Mr. Martin J. Ruggles, Mining Engineer, Anaconda Copper Mining Co., Fertilizer Department, Conda, Idaho, for their assistance in preparing sections and drawings.

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Anaconda Phosphate Plant, Beneficiation and Treatment of Low Grade Idaho Phosphate Rock

By R. J. CARO*

The Anaconda phosphate plant was put into operation in the fall of 1923. Its present daily operating capacity is approximately 170 tons of treble superphosphate and 16 tons of phosphoric acid analyzing 42 pct P_2O_5 and 52 pct P_2O_5, respectively. As a result of the continued increased demand for phosphates, Anaconda decided in 1947 to expand the plant's daily output to 270 tons of treble superphosphate and 54 tons of phosphoric acid. The present daily plant intake is 335 tons of wet (footwall) phosphate rock analyzing 31.7 pct P_2O_5 and 0.28 pct V_2O_5. The rock is shipped from Anaconda's mine at Conda, Idaho, in open railroad cars, having been crushed to pass ¾ in. mesh at the mine.[1] The expansion program will require an increased intake of 300 tons of rock per day. The hanging-wall rock analyses approximately 30.5 pct P_2O_5 and 0.11 pct V_2O_5, and will be beneficiated in a new washing plant under construction

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* Superintendent, Phosphate Plant, Anaconda Copper Mining Co., Anaconda, Mont.

[1] References are at the end of this paper.

FIG 1—Lurgi filter belt.

FIG 2—Treble-superphosphate setting belt.

in the main phosphate plant building at Anaconda.

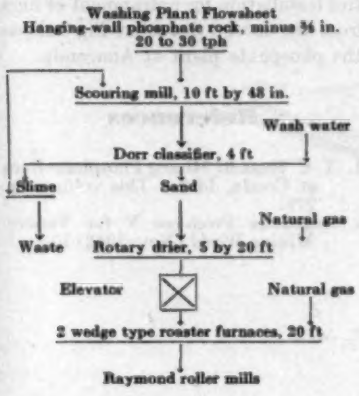

FIG 3—Anaconda phosphate plant at Anaconda, Mont.
Photograph by courtesy of Mining World.

The function of the scouring mill is to break up clay masses and to free the fine-grained shale particles having a high alumina and iron but low P_2O_5 content. This mill will operate at 15 rpm and will be equipped with interior lifts. In the washing operation, approximately 1.17 tons of hanging-wall rock of 30.7 pct P_2O_5 grade will produce 1.0 ton of sand of 33.7 pct P_2O_5 grade, giving a P_2O_5 recovery of 93.8 pct.

Table 1 . . . Typical Results Obtained Washing Conda Hanging-wall Rock

Per Cent	Crushed Rock	Sand	Slimes
P_2O_5	30.50	33.70	13.10
Al_2O_3	2.09	0.92	8.50
Fe_2O_3	1.37	0.74	4.60
SiO_2	13.00	7.90	39.80
H_2O	1.40	16.00	94.00

The scouring mill discharges at 70 to 80 pct solids. Control is based on the maintenance of a slime pulp density at about 1.04 to 1.06. Fresh water is put on deck of classifier to promote washing of the slimes from the sand. Due to packing of sand in bins, moisture content of sand is reduced from 16 to 6 pct in rotary drier ahead of calcining furnaces.

The crushed, undried "footwall rock," as received from Conda, is first calcined at a maximum temperature of 1675°F in four wedge-type, gas-fired roaster furnaces in order to burn off the excess organic material (9.0 pct). The beneficiated rock will receive the same treatment. Two additional 20 ft wedge-type furnaces will be put in operation for this purpose. The calcined rock will be ground to pass 93 pct through 100 mesh in three No. 6669 low side Raymond roller mills. Each of these mills has a capacity of approximately 12 tons of rock per hour.

The high vanadium (footwall rock) product from the roller mills goes to two Pachuca-type agitators, 15 ft in diameter and 30 ft high, operated in series. In the agitators the rock is mixed with 60° Be sulphuric acid together with weak phosphoric acid wash solution from the Lurgi filters (Fig 1). Phosphoric acid and gypsum are produced. The phosphoric acid contains vanadium compounds in solution. Phosphoric acid slurry from the agitators is passed over three 5 meter improved type Lurgi filters, where gypsum cake is washed and sluiced to waste. The Lurgi filtrate, having 27° Be strength, goes to five single effect Swenson evaporators.

Two products are obtained from the evaporators, one contains approxi-

mately 46 pct P_2O_5 and the second 52 pct P_2O_5. The acid products are pumped to six 250-ton capacity cone-bottom tanks for removal of fluosilicate salts and clarification of the acids. Clarified acid is sent to the vanadium recovery plant for the removal of this element in the form of vanadium pentoxide.[2]

The devanatized 52 pct P_2O_5 acid is shipped to consumers in rubber-lined steel tank cars, and is used as fertilizer through application to the soil by titration into the irrigation water. The devanatized 46 pct P_2O_5 acid is mixed with the low vanadium (hanging-wall rock) product from the roller mills in a model 104, 2-ton Pratt mixer. Discharge from the mixer drops onto a 48 in. rubber setting belt, 529 ft in length and traveling at a speed of 8 fpm (Fig 2), the wet material remains on the belt for 29 min during which time it takes an initial set. The setting belt discharges into a 55 by 167 ft curing shed, equipped with a traveling crane and clamshell bucket. Curing continues for 20 days, during which time the chemical action raises the available P_2O_5 content of the treble superphosphate and improves its physical characteristics.

The cured treble superphosphate from the shed is then dried in four wedge-type gas-fired roaster furnaces, after which it is crushed in 2 H. R. Sturtevant ring-roll grinders in closed circuit with Tyler Hummer-type vibrating screens equipped with screen cloth having an opening of ⅛ in. The undersize goes through a 6 by 10 ft rotating drum equipped with lifts for dedusting. A rapid stream of air is drawn through the cascading material and essentially all of the minus 100 mesh product is removed and caught in 110 wool bags which are 18 by 30 ft. These fines are then pelletized by moistening with water in a 6 by 28 ft revolving drum and dried in a 6 by 40 ft gas-fired brick-lined drier, after which they pass through a 3.5 by 15 ft cooler and are sized over a Jeffery screen, the fines go to finished product and the oversize is returned to the ring-roll grinders.

The final product is shipped to consumers in both bulk form and in 100 lb 5-ply paper bags. A screw feed device designed at Anaconda is used for the rapid loading of cars in bulk shipments. A model "A" Bagpacker fills, sews, seals, and conveys 15 bags per minute when bagged shipments are made.

In order to augment the present plant production of 235 tons of 60°Be sulphuric acid per day, a 150 ton Chemical Construction Co. vanadium catalyst contact acid plant has been constructed adjacent to the phosphate plant roasters. Iron sulphide floatation concentrates, containing about 42 pct sulphur, will be obtained from the copper concentrator tailings, and calcined in five wedge-type roasters to produce the SO_2 gas. Calcines will be sluiced to waste.

With only minor changes all additional equipment units required to meet the demands of the expanded program will be duplicates of equipment now in operation. These additions include two Pachuca-type agitators, two Lurgi filters, five Swenson evaporators, eight 250-ton capacity wood-stave, lead-brick lined acid clarification tanks. A 2-ton Pratt mixer, one 48 in. setting belt 529 ft long, curing shed with traveling crane, two 20 ft wedge-type drying furnaces for treble-super-phosphate, a No. 2 Sturtevant ring-roll grinder, deduster, and a Sly dust control installation for entrapment of fines from the second deduster. Fig 3 shows the phosphate plant at Anaconda.

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2. Anaconda Produces V for Victory. *Mining World* (Nov. 1942) 11.

Surface Strip Phosphate Mining at Leefe, Wyoming, and Montpelier, Idaho

By D. L. KING*

The San Francisco Chemical Co. has been actively interested in phosphate mining since 1908. It was, in fact, the first company to make claims on properties in the western phosphate belt. From the period of 1908 to 1947, the company's phosphate operations were centered at their Waterloo property, five miles east of Montpelier, Idaho. A small tonnage of phosphate rock was shipped annually from their underground mines in that area. In 1945, the company started the first western open-pit phosphate mine on the slopes of Waterloo Hill. In 1947, they obtained an option and lease on the Beckwith Hills' deposits in southwestern Wyoming, some 25 miles south of Cokeville. While the operations at Montpelier and the Beckwith Hills are both open pit, there are many differences which suggest separate treatment in this paper.

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* San Francisco Chemical Co., Montpelier, Idaho.

Waterloo Hill Operations

Structurally, the Waterloo Hill is the west slope of an anticline. The beddings dip from 20° to 35° to the west. Erosion has caused the footwall exposures (Well's formation) to appear in the upper areas as a series of apices, giving the outcrops a festooned arrangement. Dip faulting with subsequent erosion has induced gullies normal to the strike and resulted in eroded areas which separate the remnants of the

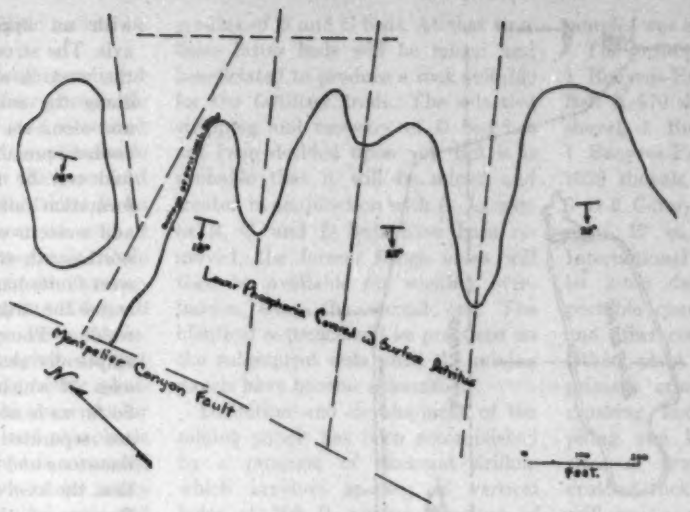

FIG 1—Plan view of Waterloo Hill showing outcrop pattern.

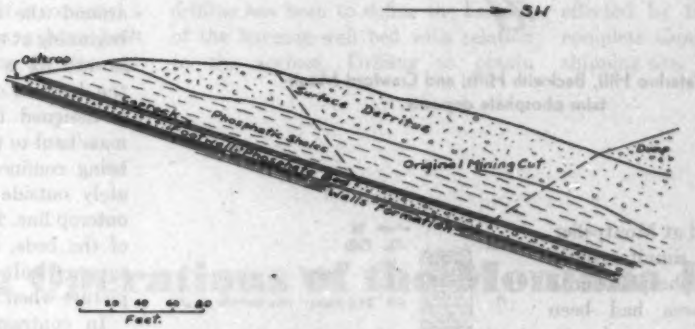

FIG 2—Sectional view of Waterloo Hill.

Phosphoria (see Fig 1). The Rex chert and several of the upper phosphatic beds are completely absent on the Waterloo Hill slope so that the overburden consists of the lower phosphatic beds together with some alluvium. Because the beds dip at a slightly steeper angle than the slope of the hill there is increased overburden as advance is made down the slope (Fig 2).

The entire hill slope area was diamond drilled first to determine the remnant and the completely eroded areas; then, once these were defined, coordinate drilling was employed with holes at 250 ft staggered pattern. Thus starting benches were determined as well as lateral spoil areas. Core recovery was very poor, but determination of horizon by drilling was clear cut due to the existence of the limestone cap rock which, fortunately, is very fossiliferous at its base and thus is an excellent marker element.

The point along the slope at which overburden could be economically removed from the footwall bedding was determined and a bench started at that point. A stripping area large enough to allow for the recovery of 20,000 tons of phosphate was stripped from the first panel. All stripping was accomplished by means of 18 to 20 yard scrapers pulled by D-8 caterpillar tractors. The first stage of stripping was carried from the grass roots to the cap rock immediately covering the footwall phosphate beds. Some ripping was necessary in order to effect the stripping of some limestone beds occurring within the phosphatic series. On low slopes "push cats" were employed to load the scrapers. As soon as an initial bench was established and a sizable area of cap rock exposed, power shovels were started along the upper edge of the exposed area to remove the limestone cap rock of fine-grained limestone, approximately 22 in. thick and sufficiently tight to preclude economical removal by any other method. The cap rock removed from the upper shovel cuts was overcast and dozed away to the spoil pile by bulldozers. As soon as sufficient area was decapped, a small power shovel was placed in operation, loading the exposed phosphate into trucks for

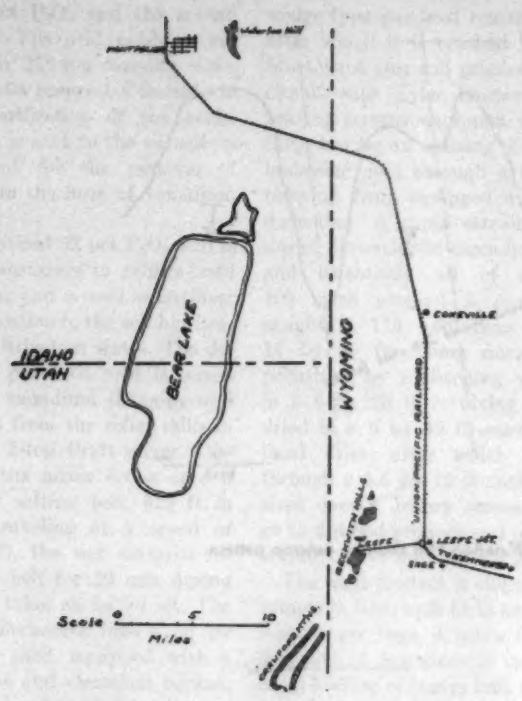

FIG 3—Waterloo Hill, Beckwith Hills, and Crawford Mountains phosphate deposits.

haulage to the rail head at Montpelier. Decapping continued simultaneously with the removal of the phosphate until the entire exposed area had been decapped. Overburden removal progressed simultaneously with these operations at the same level and adjacent to the initial bench. As soon as the initial panel had been decapped and the phosphate removed, stripping was begun in the area immediately above the panel. This procedure, which makes possible the wasting of subsequent panel overburden into the mined out area, was followed until the upper limit or the east outcrop was reached.

In twenty months of operation, beginning in 1945 and continuing until 1947, 650,000 tons of commercial grade phosphate (31.5 pct P_2O_5) was extracted from the Waterloo Hill by the above mining practices.

Wyoming Operations

The Wyoming operations of the company are centered in the Beckwith Hills area about 3½ miles west of Sage, Wyo., (a station on the Union Pacific main line about 25 miles south

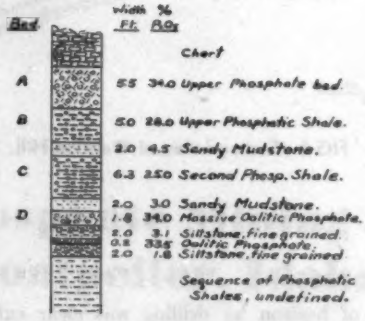

FIG 4—Generalized section of upper phosphatic shale sequence at Leefe, Wyo.

of Cokeville, Wyo., and 25 miles west of Kemmerer, Wyo.). A railroad spur was constructed to connect the mill and camp site with the Union Pacific main line about one mile west of the Sage, Wyo., station. This junction, as well as the mill and camp site, is known as Leefe, Wyo. (Fig 3).

The Beckwith Hills are a low lying end-phase of the Crawford Mountains, which appear in bold relief some three miles to the south. The Hills are part of the western limb of a gentle syncline, with an approximately north-south axis. The structure has a slight dip to the west. A sharp strike fault occurs along the eastern edge of the Hills, and along the western edge the partly eroded remnants of a tight overturned fold can be found. The Phosphoria formation outcrops along the eastern and western edges of the hills with a north-south strike. A system of east-west faults has induced erosion gullies across the strike of the beds and cut the residual Phosphoria into a series of elliptically shaped sections, the major axes of which are east-west (Fig 4). So far as is now known, these sections are separated by areas of footwall limestone and quartzite (Weber formation, the local correlation of the Wells). Because of this erosional effect, the mining panels are patterned to the residual sections, around which the outcrops of the top bed of the Phosphoria can be easily traced. Present mining practice is to strip the overburden around the fringes of each section beginning at the outcrop and extending inward for a distance determined by the depth of overburden. This practice is designed to reduce the center of mass haul to the minimum, all spoiling being confined to the areas immediately outside and below the natural outcrop line. Due to the gentle dipping of the beds, this practice produces a symmetrical stripping and mining picture when viewed aerially.

In contrast to the other western phosphate operations, the uppermost phosphatic bed is the first objective (locally, this bed is spoken of as "hanging-wall bed"), all other active western operations have as their primary objective the "footwall bed." The phosphate rock occurring in this upper bed is distinctly different from all other western productive beds in that it is a light colored, massive high-grade rock containing from 33.5 to 34.7 pct P_2O_5. Its light color, contrasted to the other western phosphate rocks, is apparently due to a long period of leaching and oxidation which has resulted in the impoverishment of the carbonaceous content which is characteristic of western phosphates. Its massive characteristics are common in this specific area (as in the Crawford Mountains to the south) due, no doubt, to severe post-deposition compression. The characteristic oölitic grain texture is apparent upon close examination, but the severe compression has resulted in nearly obscuring the individual oölites to such a degree that in polished

section only can the oölitic texture be recognized.

Directly below the hanging-wall or *A* bed lies a sequence of four phosphatic shale beds designated and described respectively as: *B* bed, a phosphatic shale member 60 in. in thickness and averaging 28.0 pct P_2O_5 (there is no parting material between *A* and *B* beds); *C* bed, a phosphatic shale member 76 in. in thickness and averaging 25 pct P_2O_5 (*C* is separated from *B* by a sandy mudstone layer 24 in. in thickness and averaging 4.5 pct P_2O_5); *D* bed, a massive oölitic phosphate rock 22 in. in thickness similar in physical appearance and phosphatic content to *A* bed (*D* bed is separated from *C* bed by a low-grade sandy mudstone 24 in. in thickness and averaging 3.0 pct P_2O_5).

Mining and recovery of the hanging-wall bed will proceed as outlined until all of the fringe areas of the various mining panels are thus stripped and mined, by which time it is expected a treatment plant will have been built and placed in operation for the upgrading of *B* and *C* beds. At that time these latter beds will be mined and beneficiated to produce a rock suitable for the fertilizer trade. The selective stripping and recovery of *D* bed has not been decided upon yet, but it is probable that it will be mined and treated in conjunction with *C*. As soon as *B*, *C*, and *D* beds have been removed, the former fringe areas will then be available for wasting overburden from the second cut. The identical sequence will be practiced on the subsequent cuts until the mining panels have become exhausted.

Definition and development of the mining panels has been accomplished by a program of diamond drilling which involves spacing of vertical holes at 200 ft centers. Because of the loose chert fragments, diamond drilling of the surface mantle was extremely difficult, and core recovery of the phosphatic beddings was very poor. The only practical value of the drilling has been to define the horizon of the hanging-wall bed with relation to the surface. Drilling to obtain samples was entirely unsuccessful.

The surface mining plant consists of: 1 Bucyrus-Erie 54-B shovel, 1 Link-Belt K-570 shovel, 1 Link-Belt K-370 shovel, 1 Bucyrus-Erie 37-B shovel, 1 Bucyrus-Erie 1030 shovel, 2 Unit 1020 shovels, 4 27FD Euclid trucks, 3 D-8 Caterpillar dozers, 2 Le Tourneau 12 cu yd buggies, 4 KBS-7 International dump trucks, 4 Chevrolet 2-ton dump trucks, 1 315CFM portable compressor, 2 wagon drills, and other complementary equipment. Other plant facilities consist of a primary crushing unit, a secondary crushing and screening unit, stock piling and loading facilities for all sizes of crushed and graded rock, crushed rock storage, and Raymond mill drying and pulverizing equipment. The plant has a capacity of approximately 150 tph of minus 1 in. crushed rock and of 25 tph of rock dried to 1 pct moisture and pulverized to 93 pct minus 100 mesh. Direct rail service is effected by the company spur, and complete camp facilities are near the shipping site.

Mining Operations of the Montana Phosphate Products Company

By R. J. ARMSTRONG,* Member AIME, and J. J. McKAY*

The Montana Phosphate Products Co. is currently operating three underground phosphate mines in Powell County, Mont.†

In this area the Phosphoria formation is from 35 to 50 ft thick and contains a bed of phosphate rock varying in thickness from 2½ to 5 ft, overlain by 12 to 30 ft of cherty quartzite, and underlain by about the same thickness of quartzite and shaly to cherty beds. In general, the phosphate bed has a reasonably hard quartzite footwall, as well as a hard cherty hanging wall, both of which greatly facilitate the mining operation.

Folding has deformed the phosphate bed to a considerable degree in the area. It is carried to surface by several of the anticlines and lowered to considerable depths in some of the synclines. The main structural features affecting the mining operations of the company are the Garrison anticline and the adjoining syncline and succeeding limb of a second anticline on the east (Fig 1). The axis of the folding strikes roughly northwest-southeast.

The three mines, namely, the Anderson, Graveley, and Luke are on different limbs of the above folds. Accordingly, the dip of the bed, and consequently the mining method, in each mine differs to a considerable degree from that of the others. For this reason, each operation will be dealt with separately.

Anderson Mine

LOCATION AND DEVELOPMENT

Topographically this mine is in a narrow valley, the bottom of which has an altitude of 5200 ft. The phosphate bed outcrops near the ridge on the east and dips towards the valley floor at an angle of from 25 to 30°,

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* Superintendent and Assistant Superintendent, respectively, Montana Phosphate Products Co., Garrison, Montana.

† See also paper by Geoffrey Gilbert: Mining Operations of the Montana Phosphate Products Company, TP 1824, *Min. Tech.* (May 1945); *Trans.* AIME (1947) 173, 240.

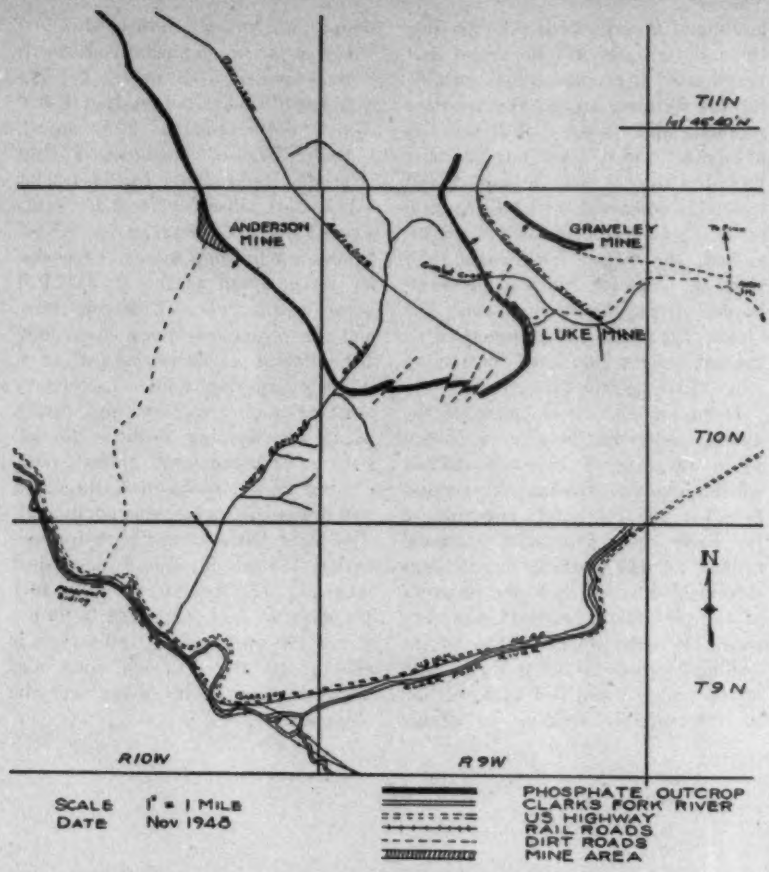

FIG 1—Location of phosphate operations, Montana Phosphate Products Co.
(Reduced approximately 50 pct in reproduction.)

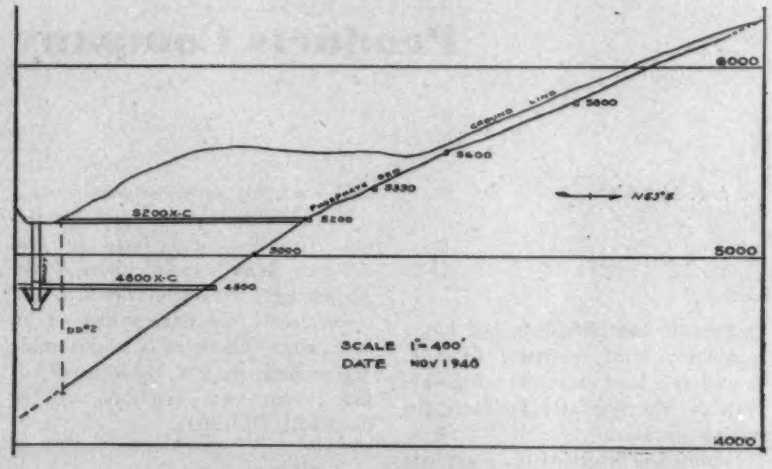

FIG 2—Cross section of phosphate bed at Anderson mine.
(Reduced approximately 40 pct in reproduction.)

which is slightly steeper than the inclination of the hillside. The strike is more or less parallel to the direction of the valley.

The mine was developed originally by tunnels driven in the hillside through the hanging-wall formation at regular level intervals. All ore above the floor of the valley was developed in this manner, but to develop the downward extension of the bed below the valley floor, it has become necessary recently to have a shaft (Fig 2). The shaft was sunk vertically through the hanging-wall formation to a depth of 475 ft, which is sufficient for the development of two levels with necessary room at the bottom for loading pockets and sumps. It is 7 by 17 ft outside the timbers and contains three 4½ by 5 ft compartments, two for hoisting and one for pipe and manway. Drifts and crosscuts are 8 by 8 ft in cross section. Drifting is done in the quartzite footwall immediately below the phosphate bed. Drift rounds are drilled with 3½ in. drifters mounted on Sullivan single jib drill carriages and mucking is done with Eimco 21 mucking machines.

STOPING

Chute cutouts are made at 72 ft centers along the drift and are driven in the phosphate bed a distance of 20 ft up the dip before the chute is installed. The chutes are of timber construction, 3 ft wide at the lip and fanned out to accommodate a 5 ft scraper at the upper end. Wingboards are installed from the upper end of the chute to the pillar on either side. Because of the flat dip of the bed, manways are not necessary and access to the stope is gained over these wingboards.

An open stope, or room-and-pillar method of mining, is used; the stopes are carried 60 ft wide and are separated by a 12 ft pillar from the adjoining stope on each side (Fig 3). These pillars are broken through at 20 to 25 ft intervals to provide better ventilation and secondary exits. Although for the most part the hanging wall is sound and stands up well, as a safety precaution two stull rows 20 ft apart with stulls at 10 ft centers are carried up the center of the stope.

The dip of the bed at 25° to 30° is too flat for the broken ore to run by gravity from the stopes to the chutes and scraping is necessary. Double drum slusher hoists of 15 hp are installed at the chutes on platforms above the drift. These operate V-type scrapers which are 5 ft long, 18 in. high, and have a capacity of about 10 cu ft. On the hoists ⅜ in. haulback and 7/16 in. load cables are used. The cables run through an 8 in. diam haulback block hung on an adjustable steel stull which can be moved readily across the face of the stope in order

that the scraper can clean out the ore across the whole width.

A stope crew consists of two miners on one shift and a scraper man on the opposite shift. As the phosphate rock is quite soft, the miners generally drill and blast the entire 60 ft face in one shift. This requires from 40 to 50 6-ft holes and on an average advances the stope face about 5 ft. Drilling is done with light 2¾ in. stopers using 1 in. quarter octagon steel and detachable bits. A set of bits will generally drill about 10 holes before requiring reconditioning. The broken rock is scraped to the chute and into the ore train by the scraper man on the opposite shift. It has been found that the economical limit for scraping at this dip, and under the described conditions, is about 300 ft. The level interval is, therefore, laid out to give this slope distance on the bed.

UNDERGROUND TRANSPORTATION AND HOISTING

Broken ore is hauled from the stopes to the shaft pockets, or in the case of levels developed by tunnels directly to the surface bunkers, in 33 cu ft Coeur d'Alene type side dump cars. Ten cars make up a train which is powered by a Mancha Titan AX, 3 ton, battery locomotive. Ore is hoisted from the shaft pockets in two 50 cu ft, side dump skips operated in balance. The skips discharge into a 400 ton surface bunker adjoining the headframe (Fig 4). From the surface bunkers the rock is transported by truck seven miles down a well-maintained, gravel road to railroad cars for shipment to the fertilizer plant.

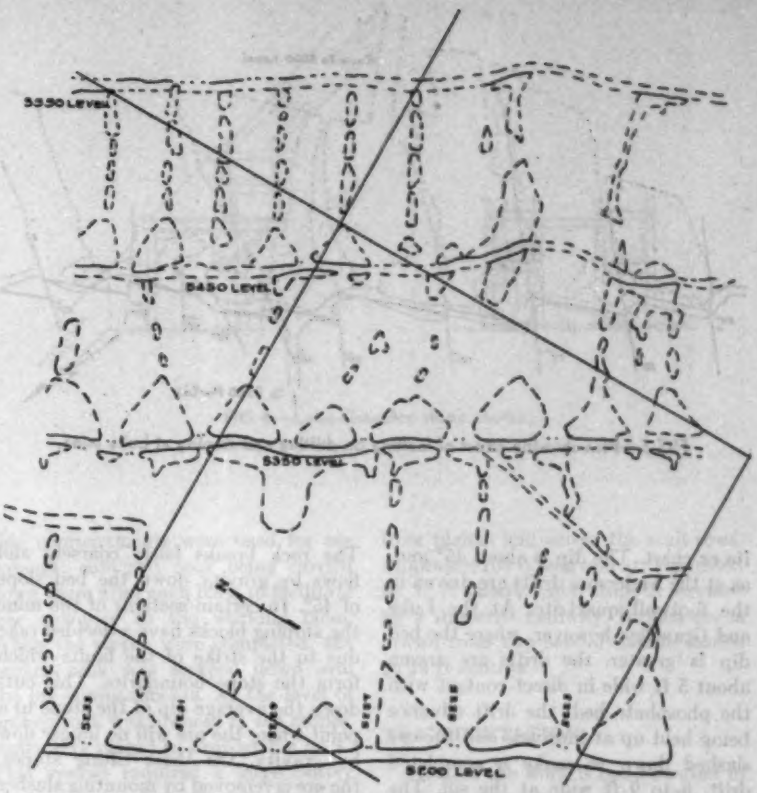

FIG 3—Plan showing room and pillar method of stoping at Anderson mine.

Luke Mine

LOCATION AND DEVELOPMENT

The Luke mine is about 6 miles southeast of the Anderson on the opposite side of the Garrison anticline. The terrain between the two mines is rugged and road communication from the Anderson is through Garrison to Avon along highway 10 and thence along the county and finally the mine road to the Luke mine, a total distance of 32 miles. A forestry-type phone line connects the Luke and Graveley with each other and with the main office at the Anderson. The Luke camp is in a thickly wooded draw which slopes quite steeply down to Gimlet Creek (Fig 1).

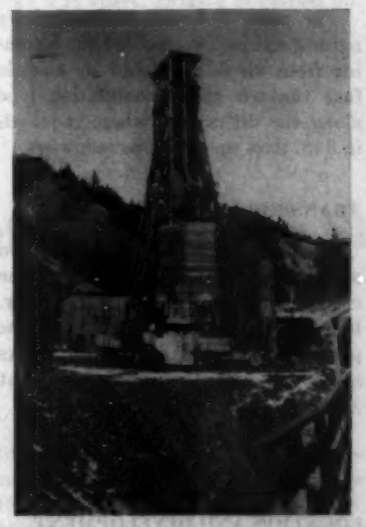

FIG 4—Anderson mine head frame and ore bin.

The Luke mine is somewhat younger than the Anderson, over half of the tonnage shipped being produced during the last two years. Development was commenced by driving a drift along the bed at 5700 ft elevation from an outcrop near the portal. All production from above this level has come through the 5700 level portal, although two short lengths of intermediate drifts have been driven at 5900 and 6050 ft elevations to service the upper workings.

In October 1946, an 8 by 8 ft crosscut adit was started near Gimlet Creek at 5300 ft elevation. This adit was driven 2700 ft and intersected the bed in July 1947. The portal is in Kootenai formation while the last 600 ft before encountering the Phosphoria formation was driven through the Ellis shales and sandstones. From a mining standpoint it is interesting to note that for the most part the Kootenai proved soft and had to be timbered while no timber was required in the Ellis. Small flows of water were encountered in nearly all the numerous sandstone beds cut by the adit, but no water was found in the Phosphoria.

The Phosphoria at the Luke consists of a phosphate bed about 3 ft wide, overlain by about 26 ft of quartz-

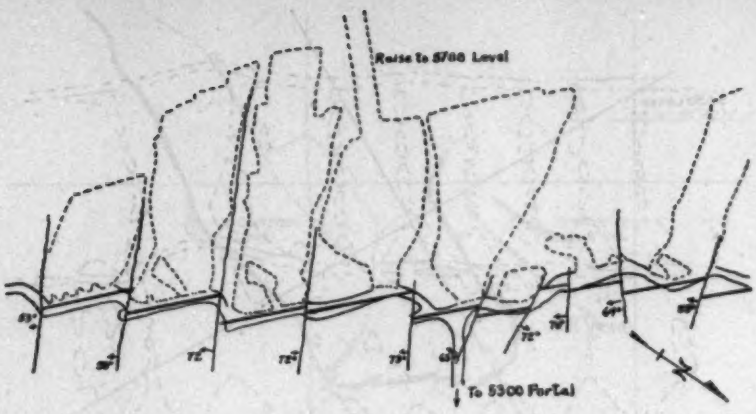

FIG 5—Plan showing effect of faulting on drifting and stoping at Luke mine.

ite or chert. The dip is about 45° and, as at the Anderson, drifts are driven in the footwall quartzite. At the Luke and Graveley, however, where the bed dip is greater, the drifts are driven about 5 ft wide in direct contact with the phosphate bed; the drift advance being held up at intervals and the ore slashed down to make a completed drift, 8 to 9 ft wide at the sill. The Luke bed is badly broken by cross faults with offsets ranging from 3 to 20 ft. These faults appear at fairly regular intervals of from 50 to 100 ft and generally displace the bed in the same direction. On encountering each fault, the drift is temporarily stopped, and, as previously stated, the ore in the back is slashed down. Chute locations are selected and sufficient rock is excavated to put in chutes (Fig 6). Unlike the Anderson, timbered manways with ladders and service slides must be carried up each stope owing to the steeper dip of the bed.

STOPING

The area between adjacent faults constitutes a single stope and is mined by a crew of two miners using a 3-in. stoper. Stoping is done in open stopes. The miners drill on planks set up on pipe extension bars called "drill pins" fashioned so as to wedge between the foot and hanging walls. No pillars are required in the stopes and normally no stulls are needed for support as the stopes are of comparatively narrow width and are separated from each other by ground left along the faults.

The ore at the Luke is harder than at the Anderson or Graveley. A set of bits will drill only two or three holes. The rock breaks fairly coarsely and flows by gravity down the bed slope of 45°. In certain sections of the mine the stoping blocks have a decided rake due to the strike of the faults which form the stope boundaries. This cuts down the average dip of the stope to a point where the ore will no longer flow by gravity. On these tilting stopes, the ore is removed by mounting slusher hoists in the manways to drag the ore from the flat parts to where it can drop down to the chutes (Fig 5).

VENTILATION

The mine is largely ventilated by natural means. On the 5300 the incoming fresh air is circulated by electric fans through 16 in. ventilation pipe along the drifts and, where required, in 8 in. lines up the stope manways.

TRANSPORTATION

Luke ore, like that of the Anderson and Graveley, is transferred from the chutes to surface bunkers in 10 car, battery motor-driven trains. From the portal it is hauled on 18 ton trucks seven miles to the railroad cars at Avon.

Graveley Mine

LOCATION AND DEVELOPMENT

The Graveley mine is about two miles northeast of the Luke on the opposite side of the Luke-Graveley syncline (see Fig 1). The strike of the bed, roughly northwest, is the same as at the Luke and Anderson, while the dip is southwest as at the Anderson. The upper part of the Graveley bed has a dip varying from 65° to vertical, while the dip of the beds in the lower level varies from 65° in the southeast to less than 45° in the northwest. Bed widths vary from 3 to 4 ft.

The Graveley ore bed was first developed by an adit driven along it from an outcrop at 5600 ft elevation. From this adit two intermediate levels were opened up at 5750 and 5900 ft elevations to service the higher stopes. A second haulage level was next opened up by drifting in at 5300 ft elevation. The surface slope flattens off below the 5300 level to such an extent that further adits would be impractical.

A winze was collared on the 5300 level, 2000 ft from the portal and sunk in the ore, bed width, and 14 ft along the strike. The winze was put down 550 ft without encountering any appreciable faulting, although the dip of the bed changed from 70° at the collar to 45° at the bottom of the winze. On completion of the winze, a skip was installed with wheels on the sides to allow for maximum capacity in the narrow bed width. The ton skip now in use is capable of hoisting 5000 tons a month, handling ore on two shifts. Owing to the difference of dip in various sections of the winze, a number of 6-in. rollers were put in on both foot and hanging walls. Water was not encountered until the last round and no serious difficulties have arisen in either sinking or operation.

Two levels are being opened up from the winze at 5100 and 4900 ft elevations. Drifting on the 5100 level is being carried on in the same manner as was done in the upper levels and consists of drifting in the footwall and then breaking down the ore from the backs of the drifts in the same sequence as that employed at the Luke. An experiment is being conducted on the lowest, or 4900, level with a different method of drifting. Advance is first made by driving a "scraper drift" in the ore, bed width, and 10 to 14 ft up the dip from the sill of the level. Then the bed is extracted for about the same distance below the sill and finally a drift of regular size is produced by excavating the waste rock from the footwall of the upper part of the slot and dropping it into the space below the sill. Broken rock throughout this operation is handled by slushers. This method has the advantage of eliminating the tramming and hoisting of all waste.

STOPING

Stope preparation, as at the Luke, consists of taking down the ore for a height of 30 ft up the bed and cutting out sufficient footwall to put in a double chute with a manway in the center (Fig 6). These chute cutouts are put in at 50 ft centers.

The phosphate bed at the Graveley is fairly regular, the most outstanding fault probably being one which dips northwest and cuts the mine roughly into two halves. This fault is filled with a clay-like material and, although it has a displacement of only 7 ft, it appears to have had a profound effect upon the nature of the ore. To the southeast of this fault the ore is fairly soft and easily drilled, while, for the most part, the ore to the northwest is relatively hard.

Several methods of stoping have been employed at the Graveley. As the bed is quite regular and has a steep dip, particularly in the upper levels, shrinkage stoping was tried out originally. It was soon found that the broken phosphate rock packed in the stope and would not run on even the steepest slopes. A modified shrinkage system was then introduced. This consisted of double chutes at 50 ft centers with a manway in the center. The whole installation was carried up with the stope advance as three timbered compartments, bed width, and each compartment 4 ft wide. The outside compartments were used for ore passage; sufficient rock being carried down them after each blast to facilitate free entrance to the working faces. When the stopes were completed, the broken rock was scraped down into the ore compartments. As the level of broken rock was lowered in the stopes, stulls were placed to support the back. This system required a large outlay for timber to carry the ore compartments and labor to empty the stopes and has, therefore, been abandoned. At present an open-stope method of mining is being successfully used with 8 in. stulls placed at 4 ft centers in rows 6 ft apart. Drilling is carried on from planks laid across the stull rows. Stopes are 100 ft long and are separated by 15 ft pillars. Each stope is serviced by a timbered manway. Broken ore is drawn from two sets of double chutes at 50 ft centers.

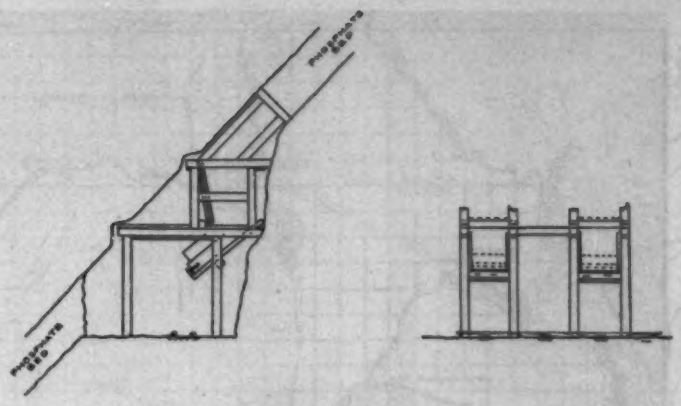

FIG 6—Luke-Graveley stope chutes.

TRANSPORTATION

Most of the work is concentrated in the area served by the winze. The ore is trammed by battery haulage to the winze pockets, hoisted to the 5300 and hauled by ore trains to the surface bunkers. From the surface bunkers, it is trucked six miles to the shipping point at Avon.

Phosphate Mining by the Simplot Fertilizer Company near Fort Hall, Idaho

By HEATH B. FOWLER*

Introduction

The surface mining operations of the Simplot Fertilizer Co. are on the Fort Hall Indian Reservation approximately 16 miles east of Fort Hall, Idaho (Fig 1). The Phosphoria formation outcrops in sec. 2, 3, 11, 14, 15, 22 and 27, T 4 S, R 37 E, Boise Meridian. The ownership of these phosphate-bearing lands is divided between individual Indians and the Bannock-Shoshone Tribe, and the royalties are paid to the respective owners on a tonnage basis, as the rock is mined. The geology and mineral resources of the Fort Hall Reservation were studied by the U. S. Geological Survey in 1913 and reported on in 1920.† However, it was not until the winter of 1945 that exploratory work was undertaken by private enterprise. Diamond drilling was begun by the Simplot Fertilizer Co. in April 1946, and stripping operations were started in June. A camp was established near

Manuscript received Dec. 23, 1948.
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† G. R. Mansfield: U. S. Geol. Survey, Bull. 718 (1920).

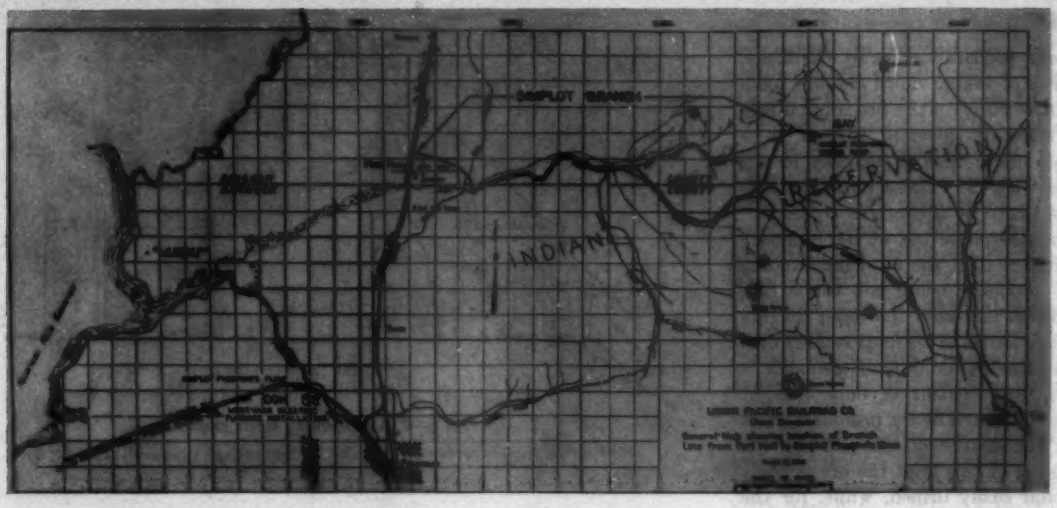

FIG 1—Index map showing location of Simplot phosphate mine at Gay, Idaho, and the Simplot phosphate processing plant at Don, Idaho.

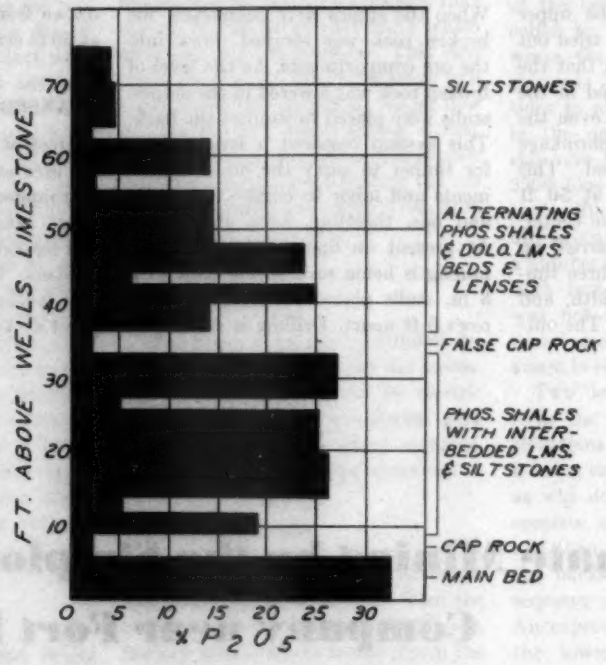

FIG 2—Formation as occurring at Simplot mine showing P_2O_5 content of lower 75 ft of Phosphoria formation.

the south end of the Phosphoria outcrop and 18 miles of oil-surfaced highway was built to provide an outlet to railroad facilities near Fort Hall. A standard gauge spur railroad line, 22 miles in length, was constructed during the summer of 1948 to connect the mine with the Union Pacific Railroad at Fort Hall. The camp buildings include a repair shop, dining hall, change room, bunk houses, offices, and a diamond drill-core library.

Geology

The stratigraphy of the Fort Hall phosphate beds is very similar to that found throughout southeastern Idaho. The general pattern and sequence of the lower 60 to 65 ft of the Phosphoria shale member, as exposed by diamond drilling, stripping and mining operations, is as follows: the bottom 6 ft of oölitic phosphate rock, locally

known as the "main bed," lying immediately above the Wells formation; 2 to 4 ft of limestone, locally known as "cap-rock"; a fairly constant 25 ft thickness of phosphatic shales and another 2 to 4 ft of limestone, which has been termed "false cap rock." Immediately overlying the false cap rock is a series of alternating phosphatic shales and dolomitic limestone beds and lenses approximately 35 ft thick. These are followed by a series of siltstones (Fig 2). The structure of the beds has been determined by a diamond drilling program which has been continuous since the beginning of operations. The major faults and fault zones, which have a definite northeast-southwest pattern, have also been determined and are a major consideration when laying out the stripping and mining panels (Fig 3). The phosphate beds generally dip 6 to 12° in a southeasterly direction and strike from north 5° east to north 45° east, giving assurance of a fairly flat-lying deposit for future strip-mining operations (Fig 4).

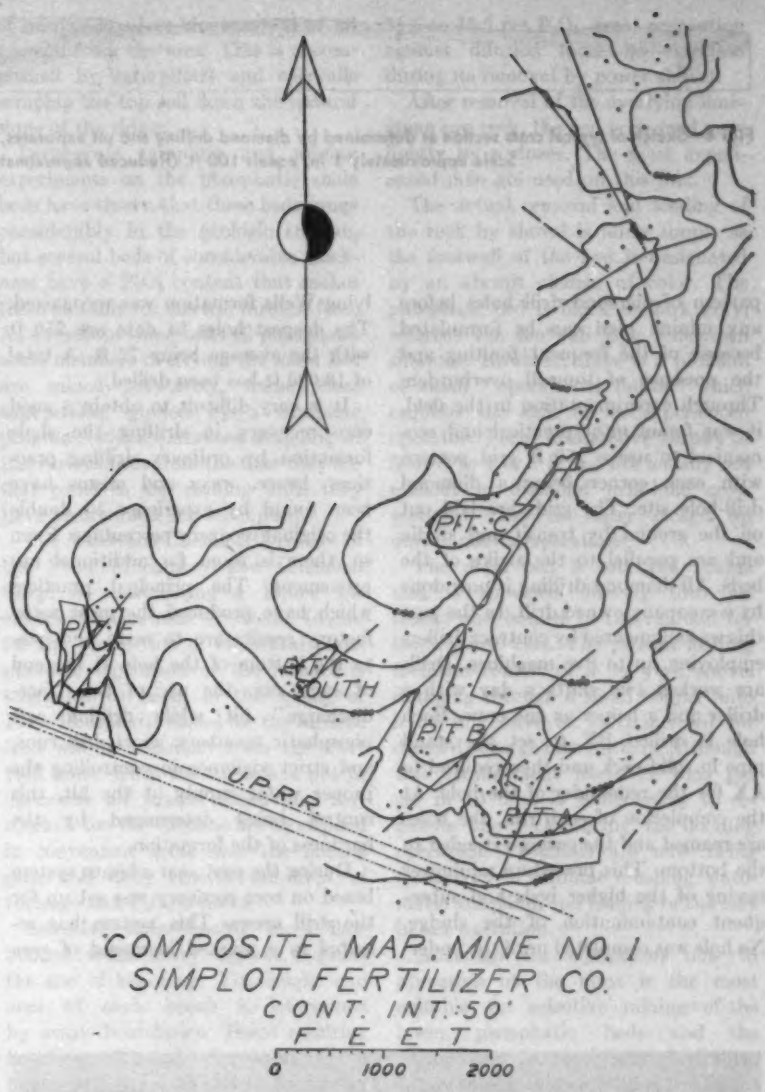

FIG 3—Plan map of south portion, Fort Hall phosphate deposits showing pit areas, diamond drill holes, and major faults.

Production

During the first year of operation at the Fort Hall mine, a total of 53,000 short tons of main bed phosphate rock was mined and hauled to Pocatello, 30 miles southwest, for processing into superphosphate. This first year's production was primarily a large scale sampling and experimental test to determine the feasibility and the economics of mining and processing phosphate rock from a virgin portion of the western phosphate field. During 1947 contracts were made with a department of the U. S. Government to furnish a large tonnage of phosphate rock for export to Japan; this contract, together with an increased expansion in the Simplot Fertilizer Company's processing requirements, necessitated a rapid expansion in the stripping and mining program in order to obtain a production for that year of 500,000 tons of phosphate rock. Additional equipment was procured for all phases of the operation. The truck road to Fort Hall was built up to a standard which would sustain trucks hauling from 15 to 30 ton pay loads. More than 100 trucks were employed in hauling to the railroad at a rate of 5 round trips per shift and 2 shifts daily. It was necessary to construct suitable detours which would stand up under such traffic while the actual road grading and black topping was in progress. The completion of the government contracts in early 1948 brought the mining program back on a more even and steady working basis and saw the beginning of selective mining and stockpiling of phosphatic shales for electric furnace feed.

The total production of phosphate rock as of Dec. 1, 1948, at the Fort Hall mine has been 786,000 tons mined and shipped. In order to uncover this amount of rock, it has been necessary to remove 900,000 yards of top soil, 155,000 yards of limestone cap rock, and 1,800,000 yards of lower-grade phosphatic shales. The approximate stripping ratio of ore to waste was one to five.

In stripping the overburden, very little blasting is required to prepare the material for shovel loading; however, a few plugs are drilled occasionally to break some large blocks of limestone. The experience of the past two years has shown that the phosphatic shales and siltstones stand well even when the pit walls are nearly vertical and up to 100 ft in height.

Diamond Drilling

It is necessary to explore the Phosphoria formation with a methodical

FIG 4—Sketch of typical cross section as determined by diamond drilling and pit exposures, showing fault blocks common at Simplot mine. Scale approximately 1 in. equals 100 ft. (Reduced approximately 50 pct in reproduction.)

pattern of diamond drill holes before any mining plan can be formulated because of the frequent faulting and the presence of top-soil overburden. Through experimentation in the field, it was found most practical and economical to use a 150 ft grid pattern with each corner being a diamond drill-hole site. The grids are laid out on the ground by transit and stadia and are parallel to the strike of the beds. All diamond drilling is now done by a company-owned drill. In the past this was augmented by contract drillers employing up to five machines. Drills are worked two shifts a day with a driller and a helper as one crew. Each hole is drilled BX to set the stand pipe in solid rock and then reduced to AX for the remainder of the hole. At the completion of each run, the holes are reamed and the casing extended to the bottom. This procedure minimizes caving of the higher beds and subsequent contamination of the sludge. No hole was completed until the underlying Wells formation was recognized. The deepest holes to date are 250 ft with the average being 72 ft. A total of 18,000 ft has been drilled.

It is very difficult to obtain a good core recovery in drilling the shale formation by ordinary drilling practices; hence, ways and means have been found by experience to double the original recovery percentage. Even so, there is room for additional improvement. The principal practices which have produced the most satisfactory results are to ream and case to the bottom of the hole at the end of each run, the use of the "face-discharge" bit while drilling soft phosphatic members, short drill runs, and strict vigilance in controlling the proper water supply at the bit, this control being determined by the hardness of the formation.

During the past year a bonus system based on core recovery was set up for the drill crews. This system has resulted in a marked increased of core recovery. All drilling is under the supervision of an engineer to ensure proper casing and sampling of each hole.

In order to overcome the discrepancies and misconceptions resulting from a lack of actual core, great emphasis has been placed on the recovery and sampling of sludges.

As mentioned previously, care is taken to prevent caving of the holes by constant reaming and casing. Hence, the sludge from each drill run is fairly representative of the footage drilled.

This sludge is recovered in sludge boxes especially made for this purpose, having large settling areas and several baffles with a small amount of fall. These permit the recovery of all material except that which is subject to sliming. These slimes are discarded and an allowance made to compensate in the subsequent sampling results.

As an added precaution against salting of the sample, water, even though rather scarce in the area, is piped and pumped up to a distance of two miles in order to prevent having to recirculate it.

The drilling program, based on the 150 ft grid pattern, brings to light the location of the faulted blocks. In several instances, holes have been collared and drilled into the fault zones and very little stratigraphic information was obtained. Whenever such a condition is encountered, the location is moved a few feet and another hole put down.

The drill rigs used by the company and the contractors were Duralite Model 1 S surface drills with a gear drive screw feed. These machines have a capacity of 800 ft with *A* rods.

Future plans call for the addition of a hydraulic feed drill rig to further improve core recovery.

FIG 5—View of strip mining operation from removing top soil by caterpillar and carryalls in background, to mining high grade rock in the foreground. Euclid dump truck in left center is stock piling furnace feed shale.

Stripping

Various methods of stripping have been tried and compared since operations began, and various time studies

and efficiency records have been kept on each piece of equipment used. These tests conducted during the last three years have resulted in the choice of the following equipment: 3 D-8 caterpillar tractors with caterpillar Model 80 scrapers, capacity 15 to 18 yards; 2 D-8 caterpillar tractors with bulldozers; 1 HD-19 Allis Chalmers tractor and bulldozer or push bell; 2 80D Northwest 2½ yard shovels; 1 37B Bucyrus Erie 1½ yard shovel; 1 Model 55 Bay City 1 yard shovel; 1 Model 25 Northwest ¾ yard shovel; 7 Model 49FD Euclid end dump trucks, 11 yards capacity; 5 Model 548 Kenworth dump trucks, 12 yard capacity; 1 Model 12 caterpillar patrol; 1 Model 212 caterpillar patrol; and 2 4500 gal water tank trucks.

Both the bulldozers and the scrapers are used in the removal of top soil. The economic maximum haul is 500 ft with the scrapers and 300 ft with the bulldozers. The 2½ yard shovels are used in conjunction with the pit trucks in removal of the phosphatic shales. This size shovel is particularly adaptable to the pit operations because of its mobility, loading speed, and capacity. The smaller shovels, ¾ to 1½ yard have proved to be better adapted to mining the high-grade phosphate rock. Their superiority is primarily due to the greater selectivity with which the operator can control the smaller shovels and thus obtain a much cleaner product for shipment.

Model 12 and 212 patrols are used to keep pit roads in good condition. This practice, in conjunction with daily sprinkling of the roads by water trucks, increases the number of loads per truck shift by 20 pct and decreases repair costs on equipment moving over dustless roads. Two D-8's with bulldozers are in the pits at all times, removing overburden or cleaning the main bed preparatory to actual mining and stripping.

Mining Plan

Prior to any strip-mining operations, mining panels are staked out on the ground as indicated by the assembled geological data. The panel size is controlled by fault blocks, depth of overburden, and the topography as it affects the location of spoils areas. After a panel is laid out, the first stage of mining involves the removal of all top soil from the area. This is accomplished by caterpillars and carryalls scraping the top soil down the natural slope of the ridges.

Large-scale sampling and mining experiments on the phosphatic shale beds have shown that these beds range considerably in the geologic column, but several beds of considerable thickness have a P_2O_5 content that makes them suitable for electric furnace feed. As a result of these tests all phosphatic shale members overlying the main bed are mined selectively, under close supervision, guided by a sampling program, which embraces sampling of the various beds from the time they are first cored in the drilling until they have been mined and stockpiled.

The Westvaco Chemical Division of the Food Machinery and Chemical Corp. is constructing a 13,000 kw electric furnace near Pocatello, Idaho, for the production of elemental phosphorous. Operation of the furnace is scheduled to begin in the spring of 1949. Phosphatic shales for the furnace feed will be obtained from the Fort Hall mine. Until the furnace is put in operation all shales mined and earmarked for the furnace are stockpiled in convenient areas near the mining pits, and daily records are kept of tonnage and P_2O_5 content.

Selective strip mining in a flat-bedded sedimentary deposit requires the use of benching. The height and area of each bench is determined by assay boundaries. These resulting benches, although appearing in a haphazard array, do have a continuity which is recognized when the whole geologic picture is known. It is the physical and chemical lensing of individual shale beds which controls the P_2O_5 content and hence the various boundaries and levels (Fig 5).

All shales which fail to contain the proper P_2O_5 for the furnace are piled conveniently and separately as a unit however, all top soil, limestone, and siltstones of very low P_2O_5 content are wasted and kept separate as another unit. Once the top soil, shales, and limestone are removed, the high-grade bed is ready for mining.

Inasmuch as it is necessary to deliver to the Simplot Fertilizer processing plant at Pocatello a rock containing a minimum of 31.5 pct P_2O_5, and as the rock in place contains from 31.5 to 33.5 pct P_2O_5, great precaution against dilution must be exercised during its removal by power shovel.

After removal of the overlying limestone cap rock, the bed is cleaned very lightly by a dozer. The most experienced men are used on this job.

The actual removal and loading of the rock by shovel is fairly simple as the footwall of the bed is designated by an abrupt change of color. The phosphate bed is black to dark gray, whereas the footwall is a red-brown siltstone. However, there is a problem of small "operational faults" which requires the skill of an experienced operator. These faults have throws of from 1 to 6 or 7 ft and are usually not evident in diamond drill-hole cross sections. They are later picked up during mining operations.

The main bed is soft and friable and is shoveled directly into pit trucks and hauled to the railroad for loading into cars. The present loading facilities consist of a ¾ yard shovel dumping through a small hopper onto a 30 in. conveyor belt, inclined for direct discharge into the cars. Although this arrangement loads 70 tons per car in 10 min, it is necessary to rehandle the rock during the loading operation. Proposals are now being considered to combine crushing, automatic sampling, and loading of the cars in one operation.

Although the equipment now in operation at the mine is the most suitable for selective mining of the lower phosphatic beds and the requirement for equipment flexibility, future mining will entail greater depths of overburden. A large proportion of this future overburden will not be amenable to selective strip mining but will be removed as units up to 75 ft in depth. These large yardages of waste material, low in P_2O_5 content, necessitate overburden removal at a low cost. Comparison studies are now being made on the larger types of earth-moving equipment.

Although Idaho has a vast reserve of phosphate rock, there is a very limited amount structurally suitable for strip-mining operations. Present day high costs and the market price of the rock, which is lagging in comparison, coupled with complex structural and stratigraphic problems require the utmost vigilance on all phases and details of a strip mining operation.

Safety in Mining at the Andes Copper Mining Company's Property, Potrerillos, Chile

By C. M. BRINCKERHOFF,* Member AIME

Safety work in mining at the Andes Copper Mining Company, Potrerillos, Chile, is divided into three parts: (1) accident prevention, (2) fire prevention and protection, and (3) silicosis prevention and control.

In the company's operations a steady campaign has been waged to reduce accidents and silicosis, and to prevent fires. In the following pages the safety record for the past twelve years will be examined and the methods explained by which the accident rate was reduced.

Table 1 contains yearly data pertaining to the scale of the mining operations and the accident record. Block caving is used for all ore production and the tonnage mined has averaged 30,000 short tons per day for the past seven years.

Table 2 classifies 44 fatal accidents which have occurred since 1935. The leading causes of accidental death are transportation of ore and use of explosives, followed in order by persons falling, falls of ground, and by asphyxiation.

In the case of serious accidents, however, the greatest cause is falls of ground, followed, in turn, by ore transportation, persons falling, explosives, and asphyxiation.

Accident Prevention

The improvement of the safety record was obtained by a combination of many factors. In surveying the results at Andes for the past twelve years, it is possible to select some features of the present operation which have contributed to the accident record. The most important of these are: (1) good staff planning, (2) good equipment, (3) an alert safety organization, and (4) education and selection of the workmen.

GOOD STAFF PLANNING

As a result of good staff planning, the mine has good working conditions, namely, good ventilation, moderate temperatures and humidity, and good lighting. These factors not only reduce accidents but also improve efficiency.

In haulage work, accidents have been reduced by having traffic move in a forward direction only. The latter makes possible a one-man train crew. Trains are centrally dispatched from loading drifts which are interconnected by signal lights, so that when a train in a lateral drift is given permission to move into a main line drift, a red light signals in all other lateral drifts in that area. Good track is maintained, and safety niches are required at all switch throws. Since safety niches were provided there have been no fatal or serious accidents at such points. All main line haulage drifts are illuminated by electric lights. Haulage drifts are kept clean. Adequate clearance has been provided between cars and between cars and the sides of the haulage drifts. The 4½ ton Granby type cars are equipped with roller bearings, and the 10 ton locomotives with trolley shoes instead of wheels. These two features are responsible for improved ore haulage conditions and contribute to greater efficiency and safety.

San Francisco Meeting, February 1949.

TP 2609 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Sept. 30, 1949. Manuscript received Dec. 2, 1948.

* General Manager, Andes Copper Mining Company, Potrerillos, Chile.

Table 1 . . . Scale of Mining Operations and Accident Record

Year	1936	1937	1938	1939	1940	1941	1942	1943	1944	1945	1946	1947
Mining Operations												
Metric tons ore mined	1,933,509	5,245,505	5,188,988	5,035,391	6,964,376	8,644,888	9,167,599	10,151,507	8,784,930	8,225,716	8,965,783	9,684,259
Lineal meters development work driven	10,350	26,460	36,565	28,736	31,109	34,568	45,023	52,266	49,166	90,192	35,014	40,525
Area undercut square meters	6,163	20,602	35,382	26,539	26,222	36,673	39,894	48,252	49,952	39,786	31,819	26,234
Accident Record per 10,000 Shifts												
Slight lost time	4.30	7.61	6.96	4.09	3.73	2.44	1.105	2.079	2.225	1.144	1.517	1.660
Serious	0.90	0.92	0.88	0.58	0.37	0.43	0.414	0.410	0.178	0.254	0.269	0.328
Fatal	0.07	0.15	0.02	0.16	0.07	0.06	0.017	0.091	0.107	0.106	0.049	0.044
Total	5.27	8.68	7.86	4.83	4.17	2.93	1.536	2.580	2.510	1.504	1.835	2.032

GOOD EQUIPMENT

In providing equipment for the miner at Potrerillos, the underlying idea has been that the best equipment possible will result in fewer men underground, a lesser risk, and greater efficiency.

The best drilling equipment available has been provided the machine man. Only electric blasting is used when blasting is necessary in ore drawing and car loading. Haulage and grizzly levels are connected by telephone for communication in such blasting work. All undercuts are blasted with Primacord, and in raise and drift work a minimum fuse length of 8 ft is used. Hot-wire lighters are used for "spitting."

Protective clothing is provided free to the miner. This includes hard-toe shoes or boots, gloves, and hard hats. The "wet" driller also receives a waterproof coat or jacket.

Another prominent cause of accidents is "falls of persons." This has occurred in ore-pass work or in work in surface pits. In this type of work the use of safety belts and ropes is mandatory and accidents are practically always the result of disobeying orders.

SAFETY ORGANIZATION

The safety department comprises a safety engineer and twelve experienced men. Ten of the latter go underground with the working shifts, inspect working faces, help the shift boss in the section to which he is assigned, especially at blasting time. The safety man is distinguished by a white helmet and is always present at the loading of undercuts and the tying-in process with Primacord. He is usually the first at accidents and summons what help may be necessary. As the hospital is close to the shaft collar, a doctor or male nurse is called underground if such services are required. A minimum of first aid is administered underground because of the proximity of the hospital.

In addition to their underground duties, the members of the safety department give talks on accident prevention at the Syndicate Hall. The safety department maintains close contact with the syndicate or union, and endeavors to see that the members are familiarized with all safety posters and circulars. At present a syndicate director is a member of the underground safety organization—a circumstance which enables the company to reach the men more easily in the safety campaign.

The safety department screens all men before they are hired and also checks the accident records, so that a man with a poor record can be eliminated from the underground force.

The safety department is also in charge of all blasting cap testing, fuse cutting and capping, and fuse and dynamite tests. All electric caps are tested for "shorts" and sensitivity in an apparatus designed by Frank W. Bell, superintendent of the electrical department. Fifty caps are tested in one operation by placing them in individual recesses, connecting wires to separate binding posts and passing 240 milliampers through each cap by a push-button control, (see Fig 1). The tested caps are then marked, repacked in boxes, and made available to the mine. About one cap in 5000 is found defective.

The safety department is in charge of ventilation, dust sampling and counting, dust control, fire prevention, and rescue apparatus and training. The underground members of the department participate in a monthly bonus and a yearly prize based on their record and contribution to the mine safety. Awards are made by a committee composed of three members of the syndicate and three of the mine staff, presided over by the mine superintendent. The main purpose of the bonus and prize system is to stimulate interest in accident prevention.

EDUCATION AND SELECTION OF WORKMEN

In the previous paragraphs the work of the safety department was mentioned in connection with the education and selection of workmen. Close cooperation is maintained between the mine staff, the safety organization, the medical department and the syndicate. Recommendations by the doctors have an important bearing on whether a man is considered suitable for underground work, or physically capable of doing the work expected of him. The mine superintendent and safety engineer discuss all accidents of a serious nature with the syndicate directors, with a view to avoiding a repetition of the accident, and to agree on proper punishment for safety rule violations.

Table 2 . . . Classification of Accidents

	Falls of Ground	Falls of Persons	Ore Transportation	Use of Explosives	Asphyxiation	Miscellaneous	Total
Fatal Accidents							
1936 to 1947, inclusive	4	6	11	10	3	10	44
Serious Accidents							
1936 to 1947, inclusive	80	19	26	19	1	112	257

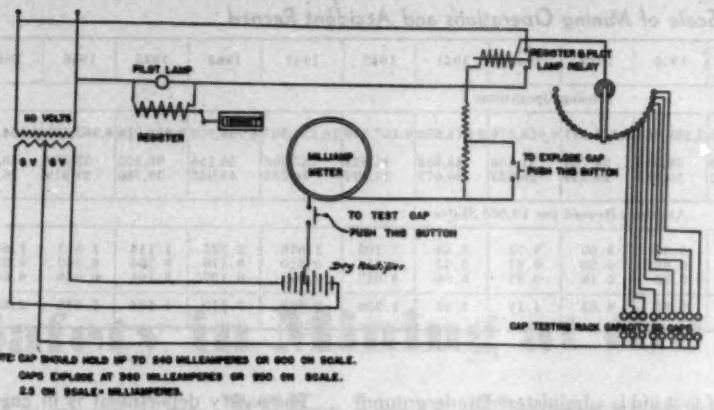
PILOT LAMP
RESISTER
110 VOLTS
6 V
6 V
MILLIAMP METER
TO TEST CAP PUSH THIS BUTTON
TO EXPLODE CAP PUSH THIS BUTTON
RESISTER & PILOT LAMP RELAY
Dry Rectifier
CAP TESTING RACK CAPACITY 25 CAPS
NOTE: CAP SHOULD HOLD UP TO 240 MILLIAMPERES OR 600 ON SCALE.
CAPS EXPLODE AT 360 MILLIAMPERES OR 900 ON SCALE.
2.5 ON SCALE = MILLIAMPERES.

FIG 1—Electric cap tester at the Andes Copper Mining Co., designed by W. F. Bell.

Fire Prevention and Protection

The safety department at Potrerillos is in charge of all fire fighting equipment and training. At present there are on hand five 2 hr McCaa and five Chemox machines. Five additional Chemox apparatus are on order. There are 10 five-man teams trained in rescue work and the use of fire fighting equipment.

For fire protection underground, the following measures have been adopted:

1. There are no buildings within 75 ft of any shaft collar or adit.

2. All shaft collars and adits have doors which can be closed manually. This is for smoke control in case of a surface fire.

3. All levels have doors close to the shafts, so that the shaft can be isolated. These doors are of steel plate construction and are of a sliding type, fitting into a slot in the side of the drift and hung from wheels which ride on a steel angle bar across the top of the drift.

4. Every shaft station is equipped with a barrel of water and a bucket.

5. All cages carry a special water hose which can be connected at each station to a special water line painted red.

6. No smoking is allowed on cages or in shaft stations.

7. No underground forges or tool sharpening are allowed.

8. All locomotive repair shops have fire extinguishers, a box of sand, and water hose and connections.

9. Electrical cables pass from the surface to all levels through raw or concreted raises.

10. The use of timber underground is kept at a minimum—at present 0.14 board foot per ton of ore. This low figure has been attained by close supervision and by the development of improved methods.

11. Special trash collections are made daily from underground lunch rooms, powder magazines, and the fuse-cutting room.

12. All areas of the mine are connected to the surface by telephone and similarly interconnected underground.

13. Ventilation plans are kept up to date.

14. Each shift boss has a copy of the ventilation plans, showing all exits and containing instructions as to the location and direction of fresh air currents in case of an emergency.

15. In case of an emergency, changes in the ventilation system may be made only by authority of the mine superintendent.

Silicosis Prevention and Control

The prevention and control of silicosis at Potrerillos has been a particularly difficult problem, due to the dryness of the climate. The small mines in the north of Chile are dry and no water is available for wet drilling. Applicants for work at Potrerillos who have worked as long as three years in the small, dry mines are usually rejected by the company doctors on the basis of their lung X-rays.

Encouraging results have been obtained by dust control measures, and by making lung examinations of applicants for work. Results of examinations for the year 1947 showed that the lung condition remained unchanged in 91 pct of the mine personnel.

The points which have contributed most to reducing the silicosis hazard and improving working conditions are:

1. Wet drilling throughout the mine.

2. Good ventilation. 500,000 cu ft of air per minute are exhausted from the mine.

3. An integrated ventilation system for the mine which is constantly being brought up to date. The underlying idea in the ventilation is to remove dirty air from mine and not to allow it to get into circulation again.

4. Wetting down haulage drifts at night.

5. Water sprays at ore dumping stations.

6. Dust control doors at ore-pass connections and wherever they are required.

7. The use of "Comfo" type respirators. In the past eleven years 13,802 were issued to the men. The wide use of the respirator has been developed with the cooperation of the syndicate and through education of the men by the staff.

8. All men are examined yearly by X-ray for silicosis or other lung disorders. The recommendation of the examining doctor is followed in cases where lung markings are indicative of trouble.

9. An alert cooperative medical staff headed by Doctor Jorge Campino.

Summary

The safety program of a mining operation is always an unfinished job. New and untrained personnel are being hired regularly. New equipment brings unforeseen hazards and problems. However, any record can be improved if an intelligent and humane attitude is taken by management and staff. Many of the safety practices now in force were started by I. L. Greninger when he was Mine Superintendent and, later, General Manager of the Andes Copper Mining Co. At present, R. L. Tobie, Mine Superintendent, Harold E. Robbins, Assistant Mine Superintendent, and Juan Abarca, Safety Engineer, are carrying on with policies and programs initiated by their predecessors.

Studies on the Activation of Quartz with Calcium Ion

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That calcium will activate quartz for flotation with anionic collectors such as soaps has been known for a number of years,[1,2,3] and the method has been applied to the concentration of various iron ores[4,5,6] as well as to other materials. Clemmer gives the permissible pH range for the activation of quartz by calcium as between 9.5 and 12.5 with an optimum value of 11.0, and Hertzog[7] has determined the calcium abstraction of iron ores and of Ottawa sand, but otherwise little quantitative data are available.

The purpose of the present investigation was to determine those conditions under which dilute solutions of calcium chloride will activate quartz for anionic flotation and to determine the amount of calcium ion adsorbed by quartz. Those variables, the influence of which was primary, such as calcium, sodium, and hydrogen ion concentrations, were studied. Other factors which influenced the results are discussed in the description of the experiments.

Three different experimental methods were used to determine the activation of the quartz. The first method was to treat a sample of ground, deslimed quartz with solutions of calcium chloride and sodium hydroxide, and to determine the calcium adsorbed by chemical analysis of the solution. Advantages are that a direct value for the amount of calcium adsorbed is obtained, and introduction of a collector which may affect the adsorption is avoided. The difficulty is to determine the extremely small differences in the calcium concentrations before and after the abstraction. The second method was to determine the activation of the quartz by observing the contact angle obtained on pressing an air bubble against the polished surface of a piece of quartz immersed in a solution of calcium chloride, sodium hydroxide, and sodium oleate. This is the well-known "contact angle" method. The third method was to determine those conditions under which an air bubble will pick up quartz particles immersed in a solution of the desired reagents.

Disadvantages common to the second and third methods are that introduction of a collector is necessary to obtain bubble contact with the mineral; very small quantities of contaminant can give erroneous results; and no information is obtainable concerning the amount of calcium adsorbed. Advantages are that tedious analyses, which at best give approximate results, are avoided, and the conditions, especially for the third method, closely simulate those of actual flotation. Although the most consistent results were obtained with the third method, this paper describes the techniques used and the results obtained from each of the three methods.

Distilled water and analytical grade reagents were used throughout except where otherwise stated. Between tests all glass articles were cleaned with a chromic-sulphuric acid mixture and sodium hydroxide.

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[1] References are at the end of the paper.

Abstraction Method

The raw material for the tests was Ottawa sand sized between 14 and 28 mesh. Batches of 500 g of this sand were dry ground for 1 hr in an Abbe porcelain mill with 2700 g of quartz pebbles. The batches were split down to samples weighing approximately 30 g, and the weight of each sample was adjusted to exactly this value.

Thorough wetting and desliming of each sample was effected in a Schoene funnel. This was found necessary for three reasons: first, the grinding produced some very fine porcelain that was capable of disturbing the adsorption equilibrium; second, it was impossible to separate the adsorbtion solution from samples containing slimes without first filtering, and tests showed that the filter materials can adsorb calcium from solution; third, the surface area of samples containing slimes is difficult to measure or even to calculate.

The weights of the deslimed samples prepared from the same batch of ground sand ranged from 22.9 to 23.2 g, indicating that the variation in size distribution between samples was slight. The wet, deslimed samples were transferred to 125 ml Pyrex Erlenmeyer flasks and the tests were carried out in these.

The samples were first washed three times with distilled water, then treated 1 hr on a mixing wheel with 0.1*N* hydrochloric acid to dissolve any activating ions present. The samples were then washed five times with distilled water and stored in distilled water for about 6 hr before being used.

For reasons subsequently explained, in some cases the samples were treated with dilute sodium hydroxide solution following the acid treatment, the pH of the wash water being adjusted with sodium hydroxide to that value desired in the adsorption test.

A sizing analysis of a representative sample of the deslimed material gave the results shown in Table 1.

Table 1 . . . Sizing Analysis of Deslimed Quartz

Mesh	Weight Per Cent
+35	0.5
−35 + 48	4.5
−48 + 65	12.0
−65 + 100	18.0
−100 + 150	16.0
−150 + 200	13.0
−200	36.0
	100.0

From Stokes' law and the known water velocity in the desliming funnel, the lower size limit of the grains in the samples was close to 10 micron This checked well with microscopic measurements of the lower size limit for the deslimed sample and with the upper size limit for the slime. Under the microscope the grains in the deslimed fraction appeared to be free from adhering slimes.

Surface area determinations were made on samples from the different grinding batches used in the adsorption tests, using the air permeability method.[8] The results were 440, 459, and 550 sq cm per gram of deslimed quartz. This method of surface determination gives only the surface area of the particles, whereas adsorption measurements give larger values because of the presence of cracks and pores. The ratio of surface as given by gas adsorption and air permeability methods on identical quartz samples is 1.8, and applying this correction to the average of the three values given above gives an average surface area of 870 sq cm per gram of quartz. That this cannot be too much in error is supported by calculations made from the sizing analysis previously given. When Ottawa sand was ground under identical conditions and the minus 200 mesh fraction was sized by sedimentation, it was found that a plot of the logarithm of size against logarithm of weight gave a straight line.[9] Using the appropriate correction factors given by Gross and Zimmerley[10] and by Gaudin and Rizo-Patrón,[11] the computed surface area of the material under investigation was 760 sq cm per gram.

Immediately before making an adsorption test, the wash water was changed and the volume adjusted so that it would be exactly 75 ml after all reagents had been added. Calcium chloride was added as a 3 g per liter solution, and sodium hydroxide as a 0.1 or 0.01*N* solution, depending upon the pH desired.

The flasks containing the quartz and reagents were revolved for the desired adsorption time on a mixing wheel at 100 rpm. The quartz then was allowed to settle for 3 min, and 50 ml of the supernatant liquid was carefully drawn off with a pipette. This portion was used for the calcium determinations, the remainder being used for a colorimetric pH determination. Blank tests were run using the same solutions but with no quartz present, and the quantity of calcium adsorbed was taken as the difference between the tests with and without quartz.

ANALYTICAL PROCEDURE

The 50 ml sample was diluted to 100 ml, two drops of bromthymol blue were added, and the solution neutralized with 1*N* hydrochloric acid plus 10 ml in excess. Then 10 ml of an 80 g per liter solution of oxalic acid was added, the solution was heated to boiling and neutralized by slowly adding dilute (1:4) ammonium hydroxide and then a few drops extra. The calcium oxalate was allowed to precipitate overnight and was filtered in a Gooch filter crucible with a filter bed of short fibered asbestos which had been leached in hot sulphuric acid (1:10) and hot ammonium hydroxide (1:4). The precipitate and beaker were washed four times with hot dilute ammonium hydroxide (1:1000). The calcium oxalate then was dissolved with 100 ml hot sulphuric acid that was poured through the filter crucible. The solution was collected in the same beaker in which the precipitation had been carried out to avoid the loss of calcium oxalate adhering to the beaker walls. The solution was heated to boiling and titrated hot with 0.05*N* potassium permanganate.

This method gave good correlation between parallel determinations. After some practice it was possible to determine calcium in amounts down to 5 mg with an accuracy of plus or minus 0.1 mg. For smaller quantities of calcium the accuracy decreased rapidly, probably because of incomplete precipitation of calcium oxalate from such dilute solutions.

As the amount of calcium adsorbed is determined as the difference between two independent analyses, the accuracy of the values given is plus or minus 0.2 mg for the whole sample, or plus or minus 0.01 mg of calcium adsorbed per gram of quartz. With microanalytical equipment for calcium determination, as described by Kirk and Moberg,[12] it should be possible to increase the accuracy substantially.

In the first determinations filter paper was used for separating the calcium oxalate, but it was found impossible to correlate parallel tests, and the determined values of adsorbed calcium were always too large. Blank tests with no calcium present demonstrated that oxalate ion was adsorbed in the filter paper from alkaline solutions, and that even excessive washing failed to remove it. However, it dissolved easily in the sulphuric acid used for dissolving the precipitate.

RESULTS

In the initial investigations only quartz samples were used which had been leached in hydrochloric acid alone. The results of the tests with these samples differed greatly from those obtained from later tests where samples treated with both hydrochloric acid and sodium hydroxide were used. The values for abstraction of calcium ions were larger and increased rapidly with the pH and increase in treatment time, contrasting with the later tests where the adsorption values were low and

rather constant for a wide pH range. The difference was greatest at high pH and characteristically the adsorption solutions were cloudy after the tests on quartz which had been pretreated with both acid and alkali. As the cloudy solution gave a Tyndall cone, it appears that colloids were dissolved in the initial tests, and the most probable explanation is that some residual slime quartz dissolved in the alkaline abstraction solution, giving a silica sol which then precipitated some of the calcium as calcium silicate. Kellogg and Vasquez-Rosas[12] claim that finely ground quartz is rather soluble in dilute sodium hydroxide solutions, and they give a solubility of 64.4 mg quartz per liter for 0.001*N* sodium hydroxide. It is obvious that even far smaller quantities of dissolved silica will affect the abstraction values of calcium very substantially.

To avoid this difficulty, the quartz samples, after acid treatment, were treated for 16 hr with sodium hydroxide solution at a pH of 12.1 and then were washed with sodium hydroxide solution of the pH desired during the adsorption tests, thus removing colloidal silica or any soluble silicates arising from residual quartz slimes. The results obtained from a 1 hr adsorption period using various pH values and two calcium concentrations are given in Table 2.

The values are given as milligrams of calcium adsorbed per gram of quartz. The accuracy of the determinations is plus or minus 0.01 mg calcium per gram of quartz, or plus or minus 33 pct for the largest value found. However, the results show the trend to be a slow increase in adsorption to a pH of about 11.0 and a fairly constant value thereafter in contrast with the rapid increase in adsorption above 11.8 found with quartz which had not been pretreated with sodium hydroxide. This indicates that silica has not dissolved in such amount as to disturb the results.

DISCUSSION

To form a close-packed monolayer of calcium when an ionic radius of 1.0×10^{-8} cm is assumed, about 50 micromols of calcium per square meter of surface is required, but it is more likely that calcium ions are adsorbed to the oxygen points in the quartz lattice. Gaudin and Rizo-Patrón[11] have calculated the number of such points from structural data for quartz, and have found a value of 4.27×10^{18} points per square meter, corresponding to an adsorbing capacity of 7 micromols per square meter. The adsorption which they found experimentally for barium ions was 10 micromols per square meter, and to explain this excess they assume that the barium ions also are adsorbed to the hydroxyl ions on the surface by exchange between barium ions in the solution and the hydrogen in the hydroxyl ions. In this way the adsorption capacity of the quartz surface under optimum conditions should double to 14 micromols per square meter.

Taking the mean of the five values found for calcium for a pH of 10.8 or higher (Table 1) it is found that 0.024 mg of calcium is adsorbed per gram of quartz. Using the average surface of 870 sq cm per gram of quartz as previously determined, this gives an adsorption of 7.1 micromols of calcium per square meter of surface, very close to the theoretically derived value.

Contact Angle Method

Several specimens of water-clear Montana vein quartz were selected for the contact angle tests. One face of each specimen was ground plane with carborundum on a glass plate. The specimens were mounted in Lucite by a standard briquetting procedure, and the exposed quartz surfaces were ground with 600 carborundum and finally polished with levigated alumina and distilled water on a polishing wheel.

To obtain fresh surfaces for the captive bubble tests, the following procedure was adopted: the specimens were wiped with a cloth moistened with sodium hydroxide solution at a pH of 12.0, polished by hand on a clean cotton pad with alumina and distilled water for some minutes, washed with distilled water and repolished on a new pad with fresh alumina and distilled water. After a thorough washing they were mounted in the bubble cell for the test run. All quartz and briquette surfaces were kept wet throughout.

Specimens treated in this way gave no contact with the air bubble when tested in distilled water, or in calcium chloride and sodium hydroxide solution. Some tests were made in which the specimens were given a final polish with alumina and sodium hydroxide at a pH of 12.0. These tests gave distinct contact angles in sodium oleate solutions for a pH ranging from 7.0 to 12.0, and the explanation is probably that some alumina dissolved as sodium aluminate and activated the quartz during the polishing. Tests run in the same way, except that minus 5 micron quartz was substituted for the alumina, gave only cling contact in the same solutions. All results described later were obtained using alumina and distilled water to avoid the activation described above.

Tests were made in the usual fashion except that the polished surface was turned down to prevent any precipitates formed from solution from accumulating on the quartz. This required that the bubble, ranging from 2 to 5 mm in diameter, be brought up to the polished surface. The values of the contact angles obtained were estimated, with an accuracy of plus or minus 5°. Solution temperature ranged between 20 and 25°C.

TEST METHOD

The specimen was suspended in the bubble cell in distilled water. Calcium chloride and sodium oleate solutions were added in that sequence, with thorough stirring between each addition. After a lapse of 3 min the specimen was tested by pressing a bubble against it for 100 sec. This was repeated on at least three places on the surface. The value of the contact angle, if any, was noted. When contact occurred, the time necessary to give contact (induction time) was determined by pressing the bubble against the surface, first for only 1 sec, and then for increasing time until a contact was obtained. The contact angle obtained in this way was also noted, and will be called the initial contact angle.

The next step was to add a small amount of sodium hydroxide and to repeat the whole procedure. Then fresh sodium hydroxide was added, and so on, so that a set of values for contact angle after 100 sec, induction time, and

Table 2 . . . Adsorption of Calcium by Quartz at Various pH Values

Calcium Concentrations	pH Value of Solution						
Mg per l	6.8	9.6	10.6	10.8	11.0	11.8	12.1
258	(0.002)	0.010	0.014	0.024	0.018	0.030	0.024
115							0.026

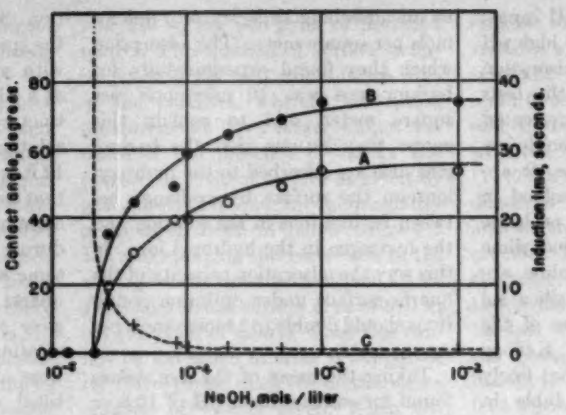

FIG 1—Contact angle and induction time as function of sodium hydroxide concentration. Sodium oleate, 20 mg per l; calcium ion, 100 mg per l. Curve *A*, initial contact angle; curve *B*, contact angle after 100 sec; curve *C*, induction time.

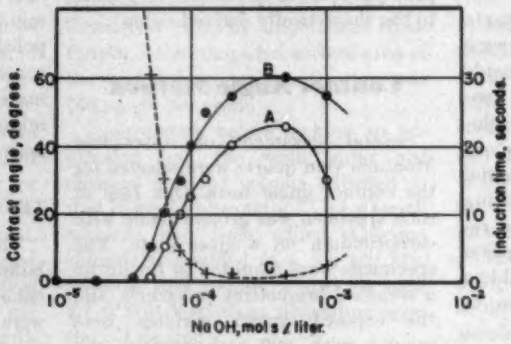

FIG 2—Contact angle and induction time as function of sodium hydroxide concentration. Sodium oleate, 20 mg per l; calcium ion, 1 mg per l. Curve *A*, *B*, and *C* same as in Fig 1.

initial contact angle was obtained for a wide range of sodium hydroxide concentrations.

In each succeeding test either the concentration of calcium chloride or sodium oleate was changed during the run, while the concentrations of the two remaining reagents were held constant. The influence of bubble age up to 100 sec was also observed, and of different reaction times before bubble application. A few tests were made in which the specimens were pretreated with a solution of calcium chloride and sodium hydroxide, and then transferred to the bubble cell which was filled with a solution of sodium oleate and sodium hydroxide.

RESULTS

When the quartz was pretreated in calcium solutions and then tested in the cell with no calcium present, a contact angle was obtained immediately after transfer, but it decreased rapidly with elapsed time, and after a few minutes no contact could be obtained. The calcium was probably desorbed under these conditions. When calcium was added to the cell together with the other reagents, the contact angle increased, and the induction time decreased in the first minutes of elapsed time. When tested after an elapsed time of 3 min or more, no observable change in the values was found. For bubble ages ranging from 1 to 100 sec, no changes in the observed values could be found.

Addition of small amounts of pine oil, up to 10 mg per liter, did not change the values of either contact angle or induction time. When a reaction time of 3 min or more was employed, tests in which the sequence of reagent addition was changed gave identical results.

The results of two tests with sodium hydroxide concentration as the variable are shown in Fig 1 and 2, of one test with changing calcium chloride in Fig 3, and of one test with sodium oleate concentration as the variable in Fig 4. The decrease in contact angle at high sodium hydroxide concentration shown in Fig 2, should be noted, and will be discussed later.

DISCUSSION

For the system of quartz in a dilute solution of calcium chloride, sodium hydroxide and sodium oleate at a pH ranging from 6 to 11, the following conclusions may be drawn:

1. Equilibrium is obtained within a few minutes, even for desorption of cal-

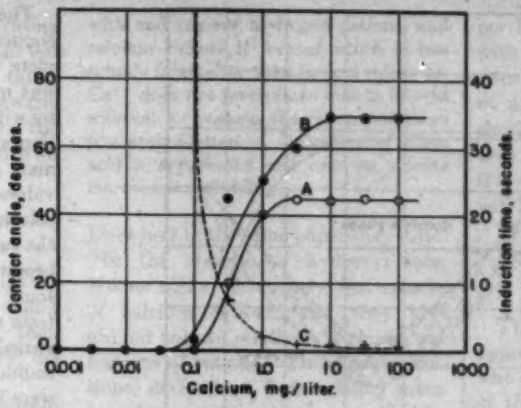

FIG 3—Contact angle and induction time as function of calcium ion concentration. Sodium oleate, 20 mg per l; sodium hydroxide, 40 mg per l. The pH changed from 10.8 to 9.8 during the test. Curve *A*, *B*, and *C* same as Fig 1.

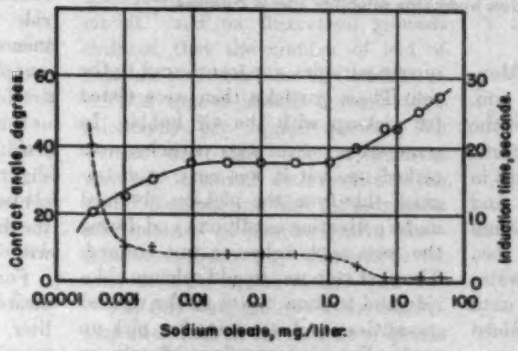

FIG 4—Contact angle and induction time as function of sodium oleate concentration. Calcium ion, 100 mg per l; sodium hydroxide, 80 mg per l; pH, 11.2. Upper curve, initial contact angle; lower curve, induction time.

cium oleate on the quartz surface.

2. Contact angle and induction time are roughly inversely proportional.

3. Under the correct conditions of pH and calcium concentration, extremely small amounts of sodium oleate, less than 0.001 mg per liter, may give contact.

The curves of Fig 1 to 4 show that in all cases a broad transition zone exists between no contact and full contact angle.

Bubble Pick-up Method

To investigate the behavior of crushed quartz particles compared with polished quartz in some of the preceding tests, a small amount of vein quartz crushed to minus 1 mm was added to the bubble cell. After each reagent addition an air bubble was pressed against the particles, and the quantity of particles that was picked up, that is, adhered to the bubble when it was lifted, was observed.

Such tests were made during the runs shown in Fig 1, 2, and 3, and the following general observations were noted: when the concentration of the added reagents was too small to give contact with polished quartz, no adherence (henceforth referred to as "pick-up") occurred. In the region where a slight contact angle was obtained, an attraction of the small particles to the bubble could be observed but no particles adhered to the bubble when it was lifted. When the reagent additions were sufficient to effect a contact angle of from 20 to 40°, with an induction time of from 10 to 5 sec, the smaller particles, about 0.1 mm diam, adhered to the bubble when it was raised, that is, a partial pick-up was observed. Finally when the conditions were such that the contact angle was from 30 to 60°, all the quartz particles could be picked up by the bubble, and the underside of the bubble could be completely coated with quartz particles. Induction times for the pick-up test were in all cases less than 1 sec.

The reagent concentrations required to give complete pick-up with the particles and a contact angle of at least 60°, with the polished quartz, and the maximum concentrations to give no pick-up and no contact, respectively, are given in Table 3.

Examination of Table 3 shows that even this crude pick-up method is more sensitive to changes in reagent concentration than the contact angle method. It was decided to investigate further the possibilities of this method, and the technique used and the results obtained are described below.

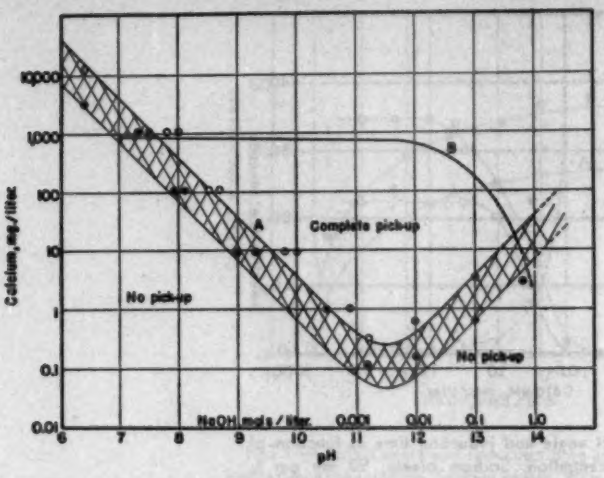

FIG 5—Pick-up for different concentrations of calcium ion and sodium hydroxide. Sodium oleate, 20 mg per l. Open circles, complete pick-up; filled circles, partial pick-up. Curve *A*, limit of complete pick-up; curve *B*, approximate limits of calcium hydroxide solubility after d'Anselme.[14]

Water-clear vein quartz from Montana was crushed dry though 1 mm, elutriated with distilled water, and the fraction plus 0.2 mm used for the tests. This fraction was leached overnight in 1*N* hydrochloric acid, washed, and leached another night with 1*N* sodium hydroxide. The material was then washed eight times with distilled water and stored under distilled water until used. Identical results were obtained when freshly leached samples and samples that had been stored for three weeks were used.

Various types of cells were constructed, and finally a simple test tube-shaped cell was adopted, consisting of a centrifuge vessel with a rubber stopper and a glass tube extending through the stopper to near the bottom. On the top of the tube was attached a piece of rubber hose closed at its upper end with a hose clamp, this hose acting as an air chamber for generating the bubbles. This design of cell can be improved, and details of a new type of cell will be given in a subsequent paper.

METHOD OF MAKING PICK-UP TESTS

The cell was filled with 30 ml of distilled water, and about 0.05 g of wet quartz particles was transferred to the cell. These particles then were tested for pick-up with the air bubble. In some cases one or two particles were picked up, but it was easy to distinguish this from the pick-up obtained under activating conditions, and during the tests such behavior was ignored. The next step was to add calcium chloride and sodium oleate in the desired quantities, and to test for pick-up again. For concentrations of calcium less than 5 g per liter, no pick-up was observed at a pH of 6.7. Then a small amount of sodium hydroxide was added, the cell was shaken and the pick-up was tested after a time lapse of 3 min. If there was any pick-up its degree was noted and the cell was then tapped several times with the finger. In some cases the particles adhered so loosely to the bubble that they became detached by this tapping, or even by the small agitation caused by the raising of the bubble in the tube. After noting the degree of pick-up after tapping, more sodium hydroxide was added, usually sufficient to double its concentration, and the pick-up test was repeated again. In this way each test run gave values for the pick-up over a wide range of sodium hydroxide concentrations.

Table 3 . . . Comparison of Pick-up and Contact Conditions

Test	Variable Reagent	No Pick-up	Complete Pick-up	No Contact	60° Contact
Fig 1	NaOH	0.00002 mol per l	0.00004 mol per l	0.00002 mol per l	0.0001 mol per l
Fig 2	NaOH	0.00005 mol per l	0.0001 mol per l	0.00005 mol per l	0.0006 mol per l
Fig 3	Ca^{++}	0.1 mg per l	1.0 mg per l	0.03 mg per l	3.0 mg per l

The concentrations at which pick-up commenced and at which it was complete were of special interest. The pH was measured at these concentrations by withdrawing about 5 ml of the solution and measuring the pH in a Coleman glass electrode pH meter. The values so obtained agreed well with colorimetric determinations, except for the ranges covered by thymolblue and bromthymolblue, which reacted in some way with the oleate ion. In some tests the concentration of calcium was varied, and the concentrations of sodium hydroxide and sodium oleate were kept constant.

RESULTS

Fig 5 summarizes the results of tests made with different concentrations of sodium hydroxide and calcium chloride. The filled points show the commencement of pick-up, the open circles complete pick-up. The values given are for pick-up after tapping the cell, as this method gave the most consistent results by eliminating the influence of slight vibrations which could not be controlled or evaluated. In most cases the difference between results with and without tapping were slight.

For the curve given the sodium oleate concentration was 20 mg per liter. Tests with 1 mg per liter gave practically the same results, but the transition from commencement of pick-up to complete pick-up was more gradual. Addition of sodium chloride had no influence on the results when the mol concentration of sodium ions present was less than from 400 to 800 times the mol concentration of calcium ions. Higher concentrations of sodium ions decreased the pick-up and when it exceeded 8000 times the calcium ion concentration pick-up was prevented almost completely.

The tests with sodium bicarbonate showed that when the concentrations and the pH are such that a precipitate of calcium carbonate is formed, there is no pick-up. Smaller concentrations of bicarbonate to some extent decrease the pick-up.

The tests with constant sodium hydroxide and variable calcium chloride concentration disclosed that for a pH between 6 and 9 the pH is very unstable, undoubtedly because of adsorption of carbon dioxide from the air admitted when the stopper was removed for reagent addition. The design of the new cell should obviate this difficulty. Care should be taken,

when a new cell is used, that activating ions released from the glass do not vitiate the results. This can be taken care of by using either Pyrex glassware, or if soft glass is used, by treating the cell with a solution of high pH until all available ions are leached out.

DISCUSSION

If it is accepted that the pick-up tests delineate those conditions under which quartz will be activated, then the following conclusions may be drawn:

1. Both hydrogen ions and sodium ions influence the activation of quartz with calcium ions. The higher the concentration of hydrogen ions and sodium ions, the more calcium ions are required to activate the quartz. From the results, the following approximate relationships were calculated:
   - *a.* There is complete activation for quartz with calcium when: the mol concentration of Ca is greater than $[H^+] \times 10^6$. the mol concentration of Ca is greater than $[Na^+] \times 10^{-3}$.
   - *b.* There is no activation of the quartz with calcium when: the mol concentration of Ca is less than $[H^+] \times 10^8$. the mol concentration of Ca is less than $[Na^+] \times 10^{-4}$.
2. When both sodium and hydrogen ions are present, then the combined effect of the ions must be taken into account, and we have the expression:

$$\text{mol Ca concentration} = 10^6[H^+] + 10^{-3}[Na^+]$$

This expression gives the minimum amount of calcium necessary to give complete activation of quartz. Curve *A* of Fig 5 represents this expression, and it may be seen that the points determined by the pick-up tests are as close to this curve as may be expected from the accuracy involved. The shaded area on Fig 5 indicates when incomplete pick-up and activation occur. The deactivating effect of sodium ions also explains the decrease in contact angle test shown in Fig 2.

Summary and Conclusions

In 1937, Fahrenwald and Newton, in a most comprehensive paper,[2] showed that:

. . . calcium ion, Ca^{++}, is a strong precipitant of quartz only in the presence of hydroxyls. Hydroxyl anions associate with and remove hydrogen cations, and calcium silicate is formed which is less soluble in alkaline than in acid solutions. Ca^{++} does not precipitate quartz in acid solution for reasons the reverse of those just stated—that is, ionization of silicic acid is suppressed and calcium silicate increases in solubility.

Elsewhere in the same paper they state: "In the absence of hydroxyl ions, H-ions resist tenaciously the attacks of calcium cations, and they give ground only to relatively high concentrations of cations." These generalizations, determined from settling rates, have been confirmed and quantified by the writers, using a modified captive bubble method.

In 1942, Gaudin and Rizo-Patrón,[11] as a result of certain experiments, concluded "that the barium ion is not abstracted by quartz from an acid circuit" and on theoretical grounds deduced that the number of loci of cationic preference on the surface of crushed quartz should amount to 7 micromols per square meter. The writers find that calcium behaves similarly to barium with respect to the first statement, and that the maximum adsorption of calcium on quartz determined experimentally amounts to 7.1 micromols per square meter.

The following picture of the adsorption of cations on the surface of quartz is proposed: all cations present in the solution tend to adsorb on the surface, and the preferential sequence is H^+, Ca^{++}, and Na^+. To displace the adsorbed hydrogen ion to any degree, the calcium concentration in the solution must be more than 10^6 times the hydrogen ion concentration, and the sodium ion concentration must be more than 10^9 times the hydrogen ion concentration. Sodium ions will commence to displace calcium ions at a concentration about 10^3 times the calcium ion concentration. As of these three ions, only adsorbed calcium ions will cause flotation with a soap collector, both hydrogen ions and sodium ions can act as depressants for calcium-activated quartz.

It should be noted that the minimum quantity of calcium necessary for activation occurs at a pH of 11.5 (Fig 5). Presumably this may be true only when sodium hydroxide is employed for pH regulation, for if some other alkali metal hydroxide were used it does not necessarily follow that the metal ion will displace calcium at the quartz surface. Actual flotation tests on quartz, using close to 75 mg Ca^{++} per liter of water, sodium hydroxide, with sodium abietate and mineral oil as collector have given 100 pct recovery at a pH of 11.5, with the recovery slowly approaching this value from the acid side, and sharply dropping once it is exceeded. The results of these tests will be given in a later paper.

Acknowledgments

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The Flotation of Quartz Using Calcium Ion as Activator

By STRATHMORE R. B. COOKE,* Member AIME

On the basis of experiments conducted on quartz using a bubble pick-up method, it was shown in an earlier paper[1] that this mineral will preferentially adsorb hydrogen, calcium, or sodium ions, depending on the relative concentrations of those ions in the solution in which the quartz is immersed. For quartz particles ranging in size from 0.2 to 1 mm, it was demonstrated that the concentration of calcium in solution (assumed present as ions) necessary to completely activate quartz for flotation is given by the expression:

$$Ca^{++} = [H^+] \times 10^6 + [Na^+] \times 10^{-3}$$

In this expression, ionic concentrations are in mols per liter. It was further shown that the pick-up method is apparently more sensitive to changes in reagent concentration than the standard captive-bubble method, and that induction times are apparently much reduced.

Since completing these earlier tests, a new type of cell has been constructed; this is shown in Fig 1 and 2. In Fig 1, *A* is a ground joint, *B* is a central tube reaching to within a few millimeters of the bottom of the cell, and *C* is a stopper which may be removed for reagent addition, and serves the further purpose of excluding carbon dioxide from the air during the test. The entire cell is constructed of Pyrex glass, and does not give the trouble experienced with the earlier cell, in which activating ions were released from the glass at high pH values. The only critical factor in the construction of the cell is the clearance between the central tube and the bottom of the cell. This clearance should be sufficiently small that the bubble can be pressed directly on the mineral grains lying on the bottom of the cell.

In the pick-up tests to be described in this paper, the reagents used were all of C. P. grade, except the sodium oleate, which was Merck's "neutral powder." The quartz employed was water-clear vein quartz, sized on screens, cleaned with both hydrochloric acid and sodium hydroxide, and given a thorough final washing with distilled water. Experimental procedures were the same as described in the earlier paper.

EFFECT OF SIZE OF QUARTZ ON PICK-UP

To ascertain the effect of particle size on the adsorption of calcium ions, the quartz was sized from minus 14 plus 20 mesh through the intervening screen sizes to minus 270 mesh plus 400 mesh. Each size was thoroughly cleaned, and then tested in the cell at different calcium chloride and sodium hydroxide concentrations, and at a constant sodium oleate concentration of 20 mg per liter.

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[1] References are at the end of the paper.

All particles, within the size range given, exhibited complete pick-up within the curve expressed by the equation above. This presumably means, when the conditions imposed by the equation are satisfied, that this maximum is independent of particle size.

However, it was found that the range through which *partial* particle pick-up occurred progressively broadened as particle size decreased. This is shown in Fig 3, in which curves *B*, *C*, and *D* show the limits at which pick-up just commences (as the pH is increased) for particles of minus 14 plus 28 mesh, minus 65 plus 100 mesh, and minus 270 plus 400 mesh size, respectively. These results indicate that for satisfactory activation, at any given pH, a lower calcium ion concentration is required for fine particles than for coarse particles.

EFFECT OF HIGH ALKALINITY ON PICK-UP

At calcium concentrations of between 1 and 10 mg per liter, and at high alkalinities, it was noticed that pick-up ceased as soon as calcium hydroxide commenced to precipitate. This effect was investigated at other calcium concentrations, with the same results. Solutions of calcium chloride, containing 10^5, 10^4, 10^3, and 10^2 mg of calcium per liter were made alkaline with sodium hydroxide until calcium hydroxide just started to precipitate, according to the following equation:

$$CaCl_2 + NaOH \rightarrow Ca(OH)_2 + 2NaCl$$

The beginning of precipitation was taken as that point at which either a faint opalescence appeared in the solution, or a Tyndall cone became ap-

parent. Referring to curve *A* in Fig 3, the triangular point at a pH of 7.0 and a calcium concentration of $10^{3.47}$ mg per liter represents the solubility of calcium chloride hexahydrate (the equilibrium form at 25°C, the temperature of the experiment). The remaining triangular points give the pH at which calcium hydroxide started to precipitate, for each designated concentration of calcium. A smooth curve can be drawn through these points. It will be noted that the transition between complete pick-up and partial pick-up agrees rather well with this curve, except for the points at 10^3 mg of calcium per liter, where the heavy calcium hydroxide precipitate obscured the pick-up tests. It seems probable that depression of quartz along this curve is due to competition between hydroxyl and oleate ions for the adsorbed calcium ions on the quartz. When hydroxyl replaces oleate ion the mineral becomes hydrophilic and is depressed.

With hydroxyl, oleate, and calcium ions present, there is the possibility of calcium hydroxide and calcium oleate forming. According to the law of mass action:

$$\frac{[Ca^{++}][OH']^2}{[Ca(OH)_2]} = K_h \qquad [1]$$

and:

$$\frac{[Ca^{++}][C_{18}H_{33}O_2']^2}{[Ca(C_{18}H_{33}O_2)]} = K_o \qquad [2]$$

Dividing Eq 1 by 2:

$$\frac{[Ca(C_{18}H_{33}O_2)][OH']^2}{[Ca(OH)_2][C_{18}H_{33}O_2']^2} = \frac{K_h}{K_o} = K \qquad [3]$$

Eq 3 states that at constant oleate ion concentration, increase in pH requires either a decrease in calcium oleate concentration or an increase in calcium hydroxide concentration, and this is in conformity with the depression of quartz along curve *A* in Fig 3. The constant *K* is independent of the calcium ion concentration, however, so that at constant pH and oleate ion concentration there should be no such closure of curve *A* with curve *E* as is shown at the low pH end of Fig 3. Either the above reasoning or the experimental results are open to question.

FLOTATION TESTS

A series of flotation tests was made to ascertain the correlation between conventional flotation and the pick-up results. Because of the unavailability at the time of massive quartz, the flotation tests were made on Ottawa sand.

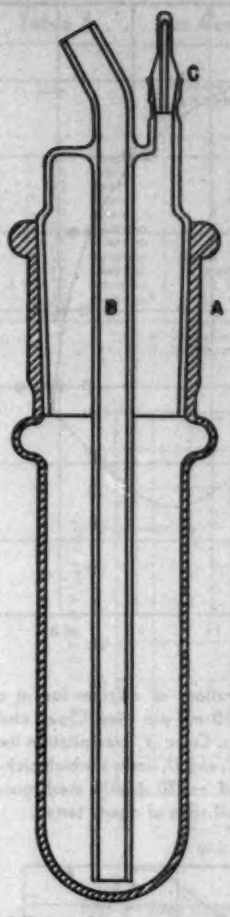

FIG 1—Cross section of cell used in pick-up tests.

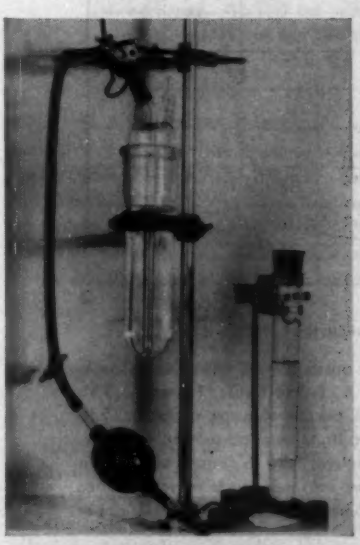

FIG 2—Pick-up cell.

The 250 g samples of sand, with a size range of minus 28 mesh, plus 48 mesh, were boiled for 10 min with 100 ml of concentrated hydrochloric acid and 50 ml of water to leach out iron and calcium. The leached sand then was washed ten times by decantation, filtered, and washed with distilled water until the washings gave no test for chloride ion with silver nitrate. Each lot was prepared for flotation by grinding in a pebble mill for 15 min at a fixed dilution with distilled water, filtering through a Buechner funnel, washing with a series of applications of boiling distilled water, and then transferring to the flotation cell. The pH of the distilled water used in grinding was 5.6, and that of the water filtered from the grind was uniformly 6.5.

The flotation cell used was identical with that described by Clemmer and Clemmons[3] except that it was fabricated of stainless steel rather than of bakelite. To maintain constant pH in the cell during a run, the froth continuously was removed, filtered, and the filtrate returned to the cell. Under these conditions, the pH at commencement and conclusion of each test differed very little except when the initial pH was 8.0 or less. Variation in pH near neutrality is attributed to carbon dioxide taken from the air. Between tests, the cell was given a number of thorough washes, first with a hot solution of sodium carbonate, and then with boiling distilled water. This treatment was always found necessary, because stainless steel seems to tenaciously retain traces of soap. Flotation procedure consisted in diluting the pulp to cell volume with distilled water, agitating, and then measuring the pH on a Coleman meter.

This was always close to 6.2, an increase in pH due probably to the effect to which Gaudin has called attention.[4] Sodium hydroxide solution was then added in sufficient quantity to give the desired pH, the cell was operated as a conditioner for 5 min, and a check pH was taken. Calcium chloride solution then was added in the required amount, the pulp conditioned for another 5 min, the pH once more measured, and the collectors, sodium abietate or sodium oleate and mineral oil, then added. After another 2 min of conditioning, the air was admitted, and flotation allowed to proceed for 8 min. No departure from these conditions occurred in the tests, the results of which will be given. All floatable mineral was removed within the 8 min. The sodium abietate was not chemically pure, and was prepared from an industrial product, Tallex.

Results of the tests are shown in Fig 4. Curve *A* shows the recovery percent-

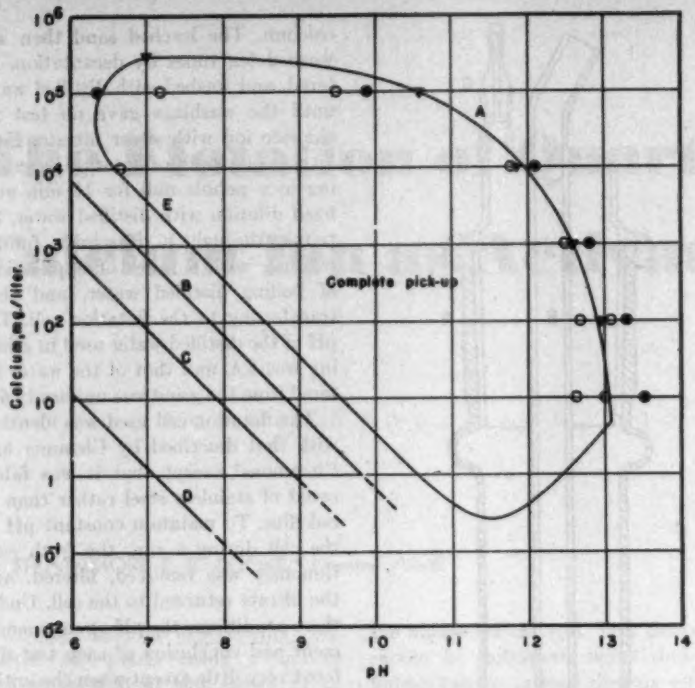

FIG 3—Pick-up of quartz for various concentrations of calcium ion at pH values above 6.0. Sodium oleate concentration, 20 mg per liter. Open circles indicate complete pick-up, filled circles no pick-up. Curve *A*, precipitation limits of calcium hydroxide (triangular points); curves *B*, *C*, and *D*, limits at which pick-up just commences for −14 + 28, −65 + 100, and −270 + 400 mesh quartz; curve *E*, limit of complete particle attachment for all sizes of quartz tested.

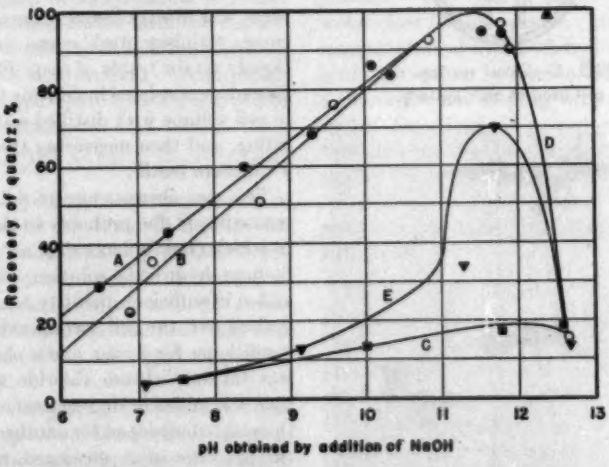

FIG 4—Per cent recoveries of clean quartz floated with calcium ion as activator at various pH values. Curve *A*, 1.0 lb $CaCl_2$, 1.5 and 3.0 lb sodium abietate, and 1.36 lb mineral oil, per ton of quartz: curve *B*, 0.5 lb $CaCl_2$, 1.5 lb sodium abietate, and 1.36 lb, mineral oil per ton of quartz: curve *C*, no calcium, 1.5 lb sodium abietate, 1.36 lb mineral oil, per ton of quartz: curve *D*, depression of calcium-activated quartz by sodium ion: curve *E*, 1.0 lb $CaCl_2$, 1.5 lb sodium oleate, and 1.36 lb mineral oil per ton of quartz.

age of quartz, using a constant quantity of calcium chloride (1.0 lb $CaCl_2$ per ton of ore, or at the dilution employed, 75 mg of calcium ion per liter of water), and both 3.0 and 1.5 lb of sodium abietate per ton of ore. Curve *B* represents the recoveries using 0.5 lb of calcium chloride and 1.5 lb of sodium abietate per ton. In all cases 1.36 lb of heavy white mineral oil was used. Curve *C* cannot be explained by calcium activation, for the quartz was treated by acid. It is possible that ferric ion is responsible, however.

Curve *E* represents the flotation of quartz with 1.0 lb of calcium chloride and 1.5 lb of sodium oleate per ton of ore and under conditions identical with those using sodium abietate as collector. No attempt was made to ascertain optimum quantity as collector. It will be noted that maximum recovery of quartz occurs at a pH of 11.5 for both curves *A* and *B* and close to this, considering the limited number of points, for curve *E*. The depressing effect of sodium ion is well shown by curve *D*, and the high alkaline part of curve *E*. It should be remembered, however, that for each change of one unit in pH, there is required a tenfold change in sodium hydroxide concentration, assuming 100 pct ionization, so that large quantities of alkali are required to attain high pH values. That depression is not due to hydroxyl ion is demonstrated by the isolated point at a pH of 12.4. In this test the quartz was floated with 1.5 and 1.36 lb per ton, respectively, of sodium abietate and mineral oil. No sodium hydroxide was used, the quartz being floated from a saturated solution of calcium hydroxide. The quartz recovery was 99.6 pct, and the calcium concentration initially in the liquid was 840 mg per liter. This would correspond to a point almost exactly on curve *A* of Fig 3.

SIZE OF QUARTZ PARTICLES FLOATED

It was noticed, at pH values other than optimum, that the froth was apparently normal, but that the particle size floated decreased rapidly with departure from a pH of 11.5. This was due not to fine particles being carried over in the interbubble liquid, but to a real decrease in the size of the particles being floated. Doubtless every flotation operator has noticed similar behavior when starvation quantities of reagent have been used.

To investigate this, sizing analyses of concentrates and tailings were made at recoveries of 21.9 and 63.5, respectively. Data from the first-mentioned test are given in Table 1, together with per cent recoveries for each size for the second test. Fig 5 shows the per cent recovery of each size plotted against size for both tests. It is obvious that at the lower recovery only fine material was floated. In both tests the recovery

in the finest product was slightly less than in the immediately preceding size fraction, due no doubt to failure of flotation in the minus 10 micron range, demonstrated elsewhere. For the test made at a recovery of 63.5 pct a sizing analysis was unfortunately not made below 400 mesh.

The results indicate that, at any given pH, less calcium ion is required in solution to activate small particles than is required to activate large particles. This confirms the results obtained with the pick-up method.

With the dilution used in the flotation cell, 1.0 and 0.5 lb of calcium chloride correspond to 75 and 37.5 mg of calcium ion per liter. Reference to Fig 3 shows that at these concentrations, *all* the quartz should have floated at pH values of approximately 8.6 and 9.0, respectively. It is obvious from Fig 4 that this did not happen. However, in the pick-up tests, the small surface of the quartz grains could have no perceptible effect on changing the calcium ion concentration in the solution. In the case of the flotation tests, however, the situation is different. Here the total surface of the quartz is large compared with the number of calcium ions available, and as the quartz adsorbs calcium ions until equilibrium is reached, calcium ion concentration in the solution will drop. Thus the curve of Fig 3 may not be directly used to predict flotation results.

From the sizing analysis given in Table 1, and using the correction factors of Gross and Zimmerley[3] and of Gaudin and Rizo-Patrón,[4] the approximate surface area of the plus 13 micron flotation feed is calculated to be 170,000 sq cm. It is difficult to assess the average size of the minus 13 micron fraction. Assuming it to be 1 micron, then with a shape factor of 1.5, the approximate surface area of the 16.63 g of material would be 564,000 sq cm, or a total of 734,000 sq cm for the 250 g of flotation feed. It is more probable that this is an underestimate of total surface rather than an overestimate.

Assuming that quartz, under *optimum conditions*, adsorbs 7.0 micromols of calcium ion per square meter, then the 250 g of flotation feed could adsorb:

$$\frac{7.34 \times 10^5 \times 7.0 \times 40 \times 10^{-6} \times 10^3}{10^4}$$

or 21 mg of calcium ion.

In a qualitative way, the rough calculation given above serves to explain why the direct flotation tests apparently do not coincide with the data of Fig 3. Another possible source of discordance is the difference between the swirling action of the flotation cell and the quiescent conditions within the pick-up cell, which would be expected to adversely affect flotation of the larger particles in the flotation cell.

Table 1 . . . Per Cent Recoveries of Quartz at Various Sizes

Size	Concentrate, Weight, g	Composite Head, Weight, g	Per Cent Recovery in Concentrate (a)	Per Cent Recovery in Concentrate (b)
− 35 + 48 mesh	0.00	0.26	0.0	0.0
− 48 + 65 mesh	0.00	2.43	0.0	3.8
− 65 + 100 mesh	0.23	21.90	1.1	7.5
−100 + 150 mesh	1.41	59.55	2.4	34.6
−150 + 200 mesh	2.74	46.25	5.9	59.4
− 74 + 52 microns	9.47	46.18	19.8	75.2
− 52 + 37 microns	9.09	23.40	38.8	91.6
− 37 + 26 microns	8.70	15.39	56.5	
− 26 + 18.5 microns	7.52	10.91	68.8	84.7
− 18.5 + 13 microns	4.92	7.15	69.0	
− 13 microns	11.41	16.63	68.8	

a. Flotation test at pH 6.5, recovery 21.9 pct
b. Flotation test at pH 9.3, recovery 63.5 pct

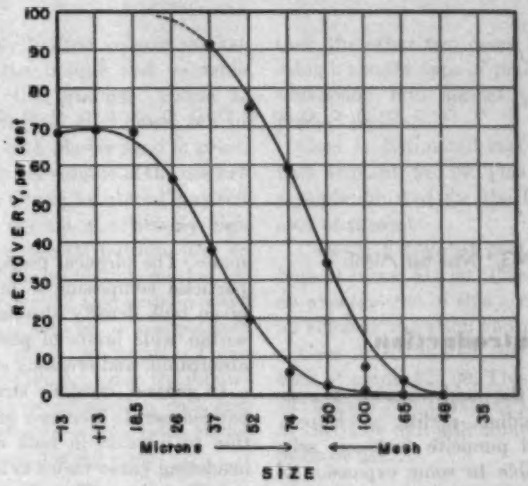

FIG 5—Size fractions in quartz concentrates prepared at pH 6.5 (recovery 21.9 pct, lower curve) and pH 9.3 (recovery 63.5 pct, upper curve), plotted against per cent recoveries for each size. At 100 pct recovery of quartz, the curve would be a straight line through the 100 pct point, parallel to the abscissa.

Summary

In a general way, the activation and flotation of quartz, using calcium ion as the activator, may be explained on the basis of exchange of hydrogen ions on the quartz and calcium ions in solution. At high sodium ion concentrations, sodium will depress calcium-activated quartz by removing the calcium. At high calcium ion concentrations, hydroxyl ions can replace collector ions and depress the quartz. Evidence is presented that fine quartz particles can adsorb calcium ions under conditions which prohibit large particles from doing the same. Because of the preliminary nature of the investigation, no attempt is made to discuss the fundamental reasons for the phenomena observed.

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Some Economic Aspects of Perlite

By C. R. KING,* Member AIME

Introduction

Most of the acid volcanic glasses such as obsidian, perlite, pitchstone, pumice, and pumicite (volcanic ash) are susceptible to some expansion if suddenly subjected to a suitably high temperature in a properly designed furnace, and subsequently suddenly cooled. Expansion results from the action of chemically combined or dissolved water expanding as a vapor within the softened heated rock particles. This expansion forms countless minute bubbles within the pasty mass of the natural glass particles. If the particles are cooled before these bubbles can escape, the resulting products are cellular, more or less spherulitized glass froths or hollow glass spheres of many types. Chemically, they are essentially aluminum silicates containing about 70 to 75 pct silica, 10 to 16 pct alumina, and 2 to 7 pct sodium plus potassium oxides. They are relatively resistant to attack by acids or alkalies, are resistant to heat up to over 1000°F, and are completely inert to insect or fungus attack.

These products are of great potential value in many branches of industry as insulation materials, aggregates, fillers, abrasives, and additives. By proper selection of the perlite rock type and control of processing conditions, expanded perlite of a bulk density varying from about 2 to 20 lb per cu ft can be made. The physical properties of the particles comprising a product of a given bulk density also may be varied within wide limits of porosity, water absorption, and crushing strength.

In general, crushing strength of expanded perlite increases in direct relation to increase in bulk density, but insulating value varies in inverse ratio to bulk density. The selection of a particular grade of expanded perlite for a given use as an aggregate is therefore always a compromise between insulating value, lightness, and friability. If porosity and water absorption are factors affecting the use of the material, factors other than bulk density are important. Some types of perlite rock tend to decrepitate excessively during expansion, resulting in a highly porous or even hygroscopic product of high friability relative to the bulk density of material produced. In general, treatment of any good perlite rock below the temperature necessary for maximum expansion, or insufficient time in the hot zone of the furnace for effective heat penetration to the core of the rock particles will result in some decrepitation and high porosity in the expanded product. On the other hand, exposure of the material to the temperature required for optimum expansion for a period of time slightly longer than that necessary for proper heat absorption will result in a slightly heavier product of extremely low porosity, i. e., a glazed surface. It is beyond the scope of this paper to describe in detail the rock types and processing methods used to produce the many types of expanded perlite in present use, or that may be produced to fill the present and future needs of industry. A few phases of the economics involved in processing and selling products made from this abundant and cheap mineral raw material, plentifully distributed in large and relatively accessible deposits in most western states and in other areas within economic shipping distance of large centers of consumption, will be treated with particular reference to the Los Angeles area.

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Relation of Bulk Density to Use and Cost of Production

Perlite differs from most other lightweight aggregates and fillers in that the raw material (perlite rock) may be quarried much as rock and sand is mined, and then shipped into consuming areas as crude rock in open gondola cars at correspondingly low freight rates and handling costs. At processing plants in or near large consuming areas the rock is converted into relatively fragile, very bulky products which in most cases are used within a short haul of the processing plant or enter into a manufactured product made at the site of the perlite processing plant.

The possible fields of use for expanded perlite and the competition within those fields from other materials divide the various types of expanded perlite into three general categories which for convenience may be based upon the bulk density of the type of material suitable for use in those fields.

Grade 1 material falls within a bulk density range of 2 to 6 or 7 lb per cu ft. It is suitable for uses where very high insulating value against heat or cold coupled with extremely light weight are important, and relatively great friability is not an objection or is an asset (as in uses involving fine particle size abrasives or fillers: pulverizing is a simple matter).

Grade 2 material falls within a bulk density range of 7 to 10 or 12 lb per cu ft. It is suitable for uses where high insulating value and relatively light weight are desirable, but where use requires sufficient particle strength to withstand mechanical mixing with a stiff binder and hand or mechanical application to a mold or surface. Hardwall plaster aggregate is at present the largest single field for use for this grade of expanded perlite.

Grade 3 material includes all expanded perlite of a bulk density over about 10 to 12 lb per cu ft. A type of material within this range would be selected for any use where light weight and high insulating value are important, but where the aggregate is subjected to relatively rough treatment in mixing and application and where relatively high compressive strength in the finished product is imperative.

The question might arise as to the reasons for arbitrarily assigning definite weight per cubic foot ranges to the three general grades of expanded perlite noted; or the reasons for picking three categories rather than more or less, since the change in physical properties corresponding to variation in bulk density is entirely gradational. The reasons for thus assigning certain ranges in bulk density to definite fields of use are largely economic. On a volume basis, the cost of producing expanded perlite increases in direct proportion to increase in bulk density. Also, as bulk density (and compressive strength of the material) increases, the competition from other cheap lightweight aggregates is keener but the potential volume of the market increases rapidly. These facts make it convenient to assign the ranges in bulk density to the three general categories as outlined.

Grade 1 material can sell at a relatively high unit price within a present comparatively limited volume market because of the unique and valuable qualities of the products. Grade 2, (plaster aggregate, etc.) must sell in competition with plaster sand in price, but the potential volume of the market is enormous. Grade 3 material may sell in a limited market at relatively high unit price for special purpose use; or may enter the large volume market for lightweight concrete aggregate in areas where it can sell on a competitive basis with other materials. These limitations are illustrated in the following estimates of production cost and possible producers selling price structure in the Los Angeles area.

Table 1 . . . Conversion Table, Cost per Ton to Cost in Cents per Cubic Foot of Expanded Perlite

Bulk Density, Lb per Cu Ft	Cu Ft per Ton	Cost, Cents per Cu Ft in Bulk at Plant, Case: (1) $15	(2) $18	(3) $25	Cost, Cents per Cu Ft in Bags at Plant,* Case: (1) $15	(2) $18	(3) $25	Grade of Material and Field of Use
2	1,000	1.5	1.8	2.5	7.5	7.8	8.5	Grade 1, special purpose material
4	500	3.0	3.6	5.0	9.0	9.6	11.0	
6	333	4.5	5.4	7.5	10.5	11.4	13.5	
8	250	6.0	7.2	10.0	12.0	13.2	16.0	Grade 2, plaster aggregate
10	200	7.5	9.0	12.5	13.5	15.0	18.5	
12	167	9.0	10.8	15.0	15.0	16.8	21.0	
14	143	10.5	12.6	17.5	16.5	18.6	23.5	Grade 3, special purpose concrete aggregate, etc.
16	125	12.0	14.4	20.0	18.0	20.4	26.0	
18	111	13.5	16.1	22.5	19.5	22.1	28.5	
20	100	15.0	18.0	25.0	21.0	24.0	31.0	

* Note: bagging cost has been allowed at 6 cents per cubic foot, which includes cost of bags, amortization on warehouse and equipment, labor and handling costs.

Estimated Possible Production Costs

The estimates in Table 1 are based on the assumption that crude perlite rock of suitable type is shipped into the Los Angeles area from the closest known deposits and processed on a scale of 50 tons per day, or 15,000 tons per year. The estimates include cost of raw material, freight charges, amortization of capital outlay over ten years, all direct costs, items of 10 pct weight loss as dust, etc., process royalty, taxes, and fixed overhead. The estimates are made on the basis of cost f.o.b. plant, on a weight (cost per ton) basis. Cost upon a volume basis may be had from the conversion table for various bulk densities. Three estimates have been made: (1) cost, with raw material cost at a minimum, for a plant integrated into a factory using the expanded perlite as an ingredient in a finished saleable article or product; (2) cost, with raw material cost at a minimum, for a plant expanding perlite for sale on the open local market; and (3) cost, with raw material cost somewhat higher and processing cost higher than the other two cases, for a plant using a special type of perlite rock for conversion into special purpose expanded perlite.

Case 1: Estimated cost per ton in bulk at plant, $15.00. This is probably a minimum cost for the Los Angeles area at present.

Case 2: Estimated cost per ton in bulk at plant, $18.00. This is probably an average cost in this area, assuming an efficient operation and process.

Case 3: Estimated cost per ton in bulk at plant, $25.00. This is probably the highest cost that may be permitted in this area and still compete in most fields of use.

Possible Future Selling Price Structure

Grade 1 expanded perlite (2 to 6 lb per cu ft bulk density): The estimates of production cost of low bulk density types of expanded perlite shown in the foregoing do not include further processing such as grinding, sizing of the material after expansion, etc. Most uses for these types of perlite fall into two categories: relatively large volume use as loose fill or preformed insulating materials such as insulating board and shapes, refrigeration and heat insulation, etc.; or relatively small volume use (small volume in the sense of each particular product using these types of perlite as an ingredient) as fillers, abrasives, filter aids, additives, etc. These last types of use usually require some sort of processing in addition to simple expansion, and might include sizing of the expanded material, grinding to fine particle size, various coatings on the particles or other special processing. In practically all cases involving the use of grade 1 expanded perlite types, marked economies are indicated if the perlite processing plant is inte-

grated into the manufacturing plant fabricating the finished article or material using these types of perlite as a major or minor ingredient. It would be difficult to predict a price structure for these types of perlite if sold by a producer on the open market. Competition would in many cases be a minor factor, and the individual small volume market as well as the unique properties of these types of perlite would in many cases result in a relatively high unit price.

Grade 2 expanded perlite (bulk density from 7 to 10 lb per cu ft): In discussing this type of perlite, the present potentially enormous market as plaster aggregate overshadows other possible uses of medium bulk density expanded perlite. From Table 1, it is apparent that packaging cost is a major item in the total cost of the low and medium bulk density material; becoming progressively less as bulk density increases. Packaging cost is particularly important in the case of plaster aggregate, which must always be moved from the producing plant to the job or building site in one form or another. So far, the only attempt to reduce packaging cost in the case of expanded perlite plaster aggregate has been the preparation of premix plaster, in which the same container carries both binder and aggregate. It is probable that in the Los Angeles area some form of returnable container would reduce packaging cost appreciably in the case of small jobs, and some form of tank truck using light detachable bodies which may be left at the job might make considerable saving in packaging cost in the case of larger jobs. If such tanks or other types of large volume containers are used, some means of easily and quickly measuring batch volumes of perlite aggregate would have to be integral with the container.

It is probable that building code and other standards will be set up in the near future governing the specifications of expanded perlite used as plaster aggregate. Among these specifications will be one governing the minimum bulk density material that may be so used. Experience so far has demonstrated that the minimum bulk density of perlite plaster aggregate at which yield is reasonably high (breakdown of the aggregate particles in mixing and application is relatively slight) is about 7 to 8 lb per cu ft. Since overall costs on a volume basis decrease in direct proportion to decrease in bulk density, the conclusion is that the bulk density of expanded perlite plaster aggregate will stabilize at about 8 lb per cu ft. This material can be produced in the Los Angeles area at a cost somewhere between 6 and 10 cents per cubic foot in bulk at plant; or between 12 and 16 cents per cubic foot in bags, f.o.b. plant. This would imply a producers selling price range between 90 cents and $1.30 per four cubic foot bag, after allowing customary producers gross markup.

In view of the appreciable weight and labor saving possible when using perlite plaster aggregate of the right type, the foregoing price range should make perlite competitive with plaster sand in this area, even if the many other advantages of perlite plaster aggregate such as high insulating value, are discounted. There is no question as to the competitive status of perlite plaster aggregate in other areas which do not have the abundant and cheap local supplies of high grade plaster sand found in the Los Angeles area.

Grade 3 expanded perlite (bulk density over 10 lb per cu ft): For most uses as structural nonbearing concrete or preformed masonry concrete aggregate, expanded perlite should cost not more than 15 to 20 cents per cubic foot ($4.00 to $6.00 per yard) in order to compete with other light aggregates on a price basis in the Los Angeles area. Reference to Table 1 shows that while these costs should be met in this area, there is little room for the customary producers markup. It seems logical, therefore, that most expanded perlite used as concrete aggregate in prefabricated shapes and structural nonbearing concrete should be processed in perlite plants at the plants manufacturing the prefabricated shapes or readymix light-aggregate concrete. The obvious advantages resulting from bulk handling of the material, waste heat utilization in other stages in the fabrication of the products, and inclusion of the cost of the aggregate in the total cost of a finished product would seem to force such a setup.

In the foregoing, the advantages of integrating perlite expansion plants into those plants fabricating products using expanded perlite in one form or another have been stressed. Most manufacturers who might use expanded perlite in their products, but have neither the technical knowledge or process rights and experience, would, while realizing the possible economies, question the practicality of integrating a perlite processing plant into their operations. To date this has been a sound objection. The perlite industry, however, is rapidly approaching the stage where a manufacturer of, say, preformed insulating shapes, can make arrangements with a firm specializing in the processing of perlite whereby an adequate perlite expanding plant may be built so integrated with the manufacturing plant that all materials are handled in bulk, waste heat is utilized, and other obvious economies are realized. Payment of a nominal royalty will ensure technical supervision and training of operating personnel, guarantee of desired capacity, uniform specification of the product, and assured supply of suitable crude perlite rock. The overall cost of expanded perlite to the manufacturer under such an arrangement would be in line with the cost estimated in this paper. Such plants may be operated economically at capacities ranging from 100 lb per hr up to many thousands of pounds per hour, with no difference in the quality of the expanded perlite produced.

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The Mining, Milling, and Processing of Perlite

By FRED D. GUSTAFSON,* Member AIME

Background

With the postwar emergency for new housing and for new industrial buildings, much research has been done on lightweight aggregates for use in concrete and plaster. The trend toward lighter weight aggregates to relieve the dead load on the higher, steel framework buildings started toward the end of the last century.

Progressively the requirements became more stringent, crushed bank slag, weighing 80 lb per cu ft as compared to 100 lb for crushed rock, was supplemented in some sections by foamed slags and expanded shales weighing from 40 to 60 lb per cu ft. In other sections cinders were used weighing from 40 to 50 lb, including sand. Sections favored with more recent vulcanism turned to pumice weighing from 30 to 60 lb, whereas other localities made use of diatomite, from 28 to 40 lb. Exfoliated vermiculite weighing from only 6 to 10 lb per cu ft came into use.

As the search for lighter weights in aggregates continued so did the search for other desirable properties such as nailability, ease of cutting or channeling, and good insulation to both sound and heat. This search eventually led to a widespread interest in the expansion of perlite, or similar volcanic glasses, with the result that in 1946, the Office of the Housing Expediter[1] listed eleven firms reporting the operation of processing plants either on pilot or commercial scale producing aggregates weighing from 2 to 16 lb per cu ft.

Geology

Perlite is one of the large family of rocks originating from a granitic or acidic magma and generally has a "rhyolitic composition with a marked perlitic structure."[2]

The name "perlite" is a derivation from "perlstein" originally given "to certain glassy rocks (hyaloliparites, hyalo-rhyolites) with numerous concentric cracks, from the fancied resemblance of broken out fragments to pearls."[3] The cause of the perlitic texture is somewhat controversial. Johannsen attributes the texture to strain set up in the glass by cooling and describes the finding of peculiar, glassy balls, called marekanité, more or less rounded and showing concave indentations, which may be left when perlite is broken. Frank Rutley, on the other hand, in 1881, while working on vitreous rocks from Montana, observed that there was incipient or partially started crystallization with segregation of water toward the uncrystallized portion in all specimens of perlite examined. Further he observed evidence of strain around focii of crystallization to which he ascribed the development of the spherical cracking which gives the rock its perlitic texture.[4]

Widespread over the western and southwestern United States where Tertiary vulcanism is exposed, perlite commonly is associated with andesitic, basaltic, and rhyolitic lavas with interbedded tuffs and ash of Eocene age although some deposits are associated with rocks assigned to the Pliocene age.[5,6] In nearly all cases the perlite overlies water laid tuff beds or agglomerates or breccias of explosive nature, an outstanding exception being the occurrence of perlite as the chilled selvage of andesite intrusions.[7,8]

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[1] References are at the end of the paper.

Table 1 . . . Perlites of Steens Mountains[5]

SiO_2	67.05	68.66
Al_2O_3	14.91	14.44
FeO	1.48	1.28
Fe_2O_3	0.92	0.80
MgO	0.65	0.18
CaO	2.44	1.96
Na_2O	4.15	3.86
K_2O	3.04	3.28
H_2O (+105°C)	4.35	4.80
H_2O (at 105°C)	0.50	0.40
CO_2	none	none
TiO_2	0.34	0.25
P_2O_5	0.12	trace
S	trace	none
MgO_2	trace	none
Totals	99.95	99.91

Analyses by W. H. and F. Herdsman.

Table 2 . . . Perlites of Mutton Mountains[10]

SiO_2	73.79	73.28
Al_2O_3	12.40	12.55
Fe_2O_3	0.52	0.58
FeO	0.62	0.63
MgO	0.11	0.08
CaO	0.80	0.80
Na_2O	3.16	2.97
K_2O	4.84	5.00
H_2O(+105°C)	3.24	3.60
H_2O(−105°C)	0.25	0.19
TiO_2	0.09	0.09
P_2O_5	0.01	0.01
MnO	0.02	0.02
Totals	99.85	99.80

Analyses by James Kerr, University of Minnesota, Oct. 23, 1946.

Available chemical analyses of perlites from the western and southwestern United States indicate that they are varieties of leucorhyolites rather than of normal rhyolites, leucorhyolites having less than 5 pct of dark minerals, normal rhyolites having more than 5 pct. This conforms with Johannsen's observation that leucorhyolites are the prevailing kind in North America.[9] Tables 1 and 2 give

analyses of samples from the Steens Mountains and from the Lady Frances mine in the Mutton Mountains, Oregon.

Comparison of these perlite analyses with analyses of obsidian and pitchstone shows very little difference in composition, the principal difference being in percentages of combined water. Table 3 gives the analyses of obsidian and pitchstone.

Table 3 . . . Analyses of Obsidian and Pitchstone[11]

	Obsidian	Pitchstone
SiO_2	73.84	70.19
TiO_2	0.14	0.07
Al_2O_3	13.00	12.37
Fe_2O_3	1.82	1.45
FeO	0.79	0.81
MnO	0.07	0.02
MgO	0.49	0.91
CaO	1.52	1.43
Na_2O	3.82	3.03
K_2O	3.92	3.57
H_2O	0.53	6.48
P_2O_5	0.01	0.03
FeS_2	0.02	
Totals	99.97	100.36

On the basis of the analyses given obsidian contains less than 1 pct, perlite from 3.24 to 4.80 pct, and pitchstone from 4.52 to 10.05 pct water. According to Johannsen the essential difference between obsidian and pitchstone is in the water content, the personal equation entering largely into the naming of the rocks, with obsidians of over 7 pct water and pitchstones of less than 2 pct water having been reported. Johannsen suggests that 3 or 4 pct of water would be a better division point.[12] It would seem, therefore, that texture alone identifies perlite insofar as the rhyolitic glasses are concerned.

The expansion of certain of the rhyolitic glasses when subjected to heat is attributed to the presence of combined water and to some extent seems to be influenced by the release of internal strain. That not all glasses being expanded are perlite is acknowledged, but as a matter of convenience henceforth they shall be referred to as perlite.

Mining

Development of Dant and Russell's Lady Frances mine, in the absence of any data other than that from open pit, surface operations elsewhere, was patterned after standard methods for metal mines. Surface test pits were dug and sampled over a large area. Work was then concentrated on the section giving the best combination of suitable perlite and availability for transportation. The result was the driving of approximately 2000 ft of drifts, crosscuts, and raises whereby approximately 500,000 tons of perlite were blocked out.

During the course of the development work occasional, seemingly unrelated bands of an intrusive resembling rhyolite, locally referred to as "jasperite," were encountered. Dips and strikes along the bands which varied in thickness from a fraction of an inch to several feet were irregular. Many lithophysae were noted on the outer faces of the bands and both the septa thereon and the flow lines within the bands were parallel either with flow lines within the perlite or with cooling fractures within the perlite, and an extremely altered zone of perlite bounded each side of the bands varying from a fraction of an inch to several inches in width. During the initial stages of production it was found that the bands became larger, and more numerous as mining progressed higher into the deposit, and since neither the jasperite nor the adjoining altered perlite would expand, even though the latter ran as high as 8 pct in combined water, it became necessary to sort and discard them insofar as possible. It further was found that the intrusive bands had accompanied post-perlite rhyolite intrusions.

Since fragmentation from blasting usually is very good in perlite the jasperite bands which break coarser are amenable to hand sorting on the basis of size as well as appearance. Currently, sorting of jasperite is accomplished at three places at the mine, all mining now being done by open pit method: (1) fragments too large to be moved by hand are removed from the pit face by bulldozers, (2) fragments too large to pass through the coarse ore bin grizzly are hand sorted by the grizzly man, and (3) fragments too large to pass through the fine grizzly ahead of the crusher partially are picked out by the crusher operator.

That similar rhyolitic bands commonly are associated with perlite deposits elsewhere has been confirmed by J. F. Foran.[a] Mr. Foran commented, however, that not until he had visited the Lady Frances mine did he realize the extent of the bands within the deposit nor the amount of alteration accompanying them.

[a] Formerly Consulting Mining Engineer for Climax Molybdenum Mining Co. Personal communication.

No timbering was found to be necessary during the driving of the 6 by 8 ft development headings, and mining costs were quite low. Drill footage per bit was as high as 162 ft. Five to six 5 ft holes were drilled to pull a minimum 7 ft round. Powder consumption is indicated by the past month's production of 1152 tons of perlite mined with only 250 lb of powder having been used.

During the underground stage of exploration and development, with an eye to the probable necessity at a much later date of mining underground after the surface perlite has been worked out, considerable test work was done in the shrinkage stope and in a later slusher stope to determine the amount of open ground that could be carried without timber. Starting at 8 ft the stopes gradually were widened to 15 ft at which point the back started arching and the sides started sloughing. The maximum permissible width was determined to be 12 ft. Tentative plans for further underground stoping, at such time as that should become necessary, specify sublevel stoping in a series of comparatively small or narrow blocks to facilitate rapid removal of pillars and retreat.

The underground phase at the mine was accompanied by a detailed sampling program. Grab samples were taken of each round blasted and 3 to 4 in. channel samples were cut at 5 ft intervals throughout all the workings, each channel sample giving a minimum of 50 lb of rock to be tested. In addition to the foregoing, special samples were taken of each type of perlite exposed, the types being listed according to textures which varied from true perlitic to splintery to more or less massive, blocky cooling fractures. An interesting observation has been made by Oliver C. Ralston who suggests that the type of cooling fracture may be indicative of the proximity of the center of eruption of the flow.[a]

Table 4 . . . Averages of Combined Water, North Drift

Samples Averaged	Combined Water,* Pct
5— 50	3.52
50—145	3.36
145—185	3.23
185—215	3.31

* In running the determination of percentage of combined water the sample is dried at 212°F for 30 min after all apparent moisture has been removed. The sample then is weighed and placed in a furnace at from room temperature to not over 300°F. The temperature gradually is raised to 1200°F at which point it is held for a minimum of 1 ½ hr. The sample is allowed to cool in a desiccator before final weighing.

Within the groups of samples averaged above, adjacent samples varied as much as 0.26 pct.

[a] Personal communication upon visit to Lady Frances mine, 1946.

Considerable variation was found in the amount of combined water within the deposit. Table 4 lists the averages of water analyses on channel samples cut along the North Drift starting at Station 5 plus 5 ft thence continuing north.

Milling

The milling of perlite is a relatively simple process compared to many of the flowsheets for basic metals or even for other nonmetallics, but, nevertheless, the need for a closely controlled end product, or concentrate, is of extreme importance to the later processing.

As will be explained later the elimination of as many fines as possible from the final expanded product is imperative. Since research has shown that a portion of the fines in the end product are the result of fines (minus 100 mesh material) in the furnace feed or mill concentrate, every effort is made to reduce the fines in the mill. On the other hand, to maintain proper feed control on the furnace as well as to insure the desired screen analysis on the end product, it is just as imperative that there not be an excess of coarse material (plus 10 mesh). Accordingly, from the primary crusher at the mine the perlite goes to storage at the mill thence to a 2 by 6 rod mill for wet grinding at minus 35 pct density. The very friable nature of the perlite due to incipient cooling fractures makes it necessary to remove the perlite as quickly as possible from the grinding circuit once it is down to size to prevent overgrinding.

Samples are taken at half hour intervals throughout the shift and are combined to form composite samples of the day's run. Screen analyses and determinations of percentage of combined water are made on each composite sample. The average of screen and water analyses for the concentrates for the period January 1 to May 15, 1948 is: plus 20, 4.6; plus 60, 73.7; plus 100, 15.4; minus 100, 6.3; percentage of water, 3.16.

During the period of the above average a 20 mesh screen cloth was in use in the mill. At the end of that period a 10 mesh screen cloth was installed which resulted in a marked decrease in the amount of fines as shown in the screen average for May 15 to July 1, 1948: plus 10, 0.3; plus 20, 28.1; plus 60, 55.8; plus 100, 11.1; minus 100, 4.7; percentage of water, 3.26.

A still better average was achieved during August when the minus 100 mesh material dropped to 3.1 pct.

The average life for the medium weight stainless steel screen cloth used is slightly over 3000 tons. The analysis for August 1948, on the screen overflow, was: plus 10, 34.6; plus 20, 53.3; plus 60, 11.5; plus 80, 0.2; plus 100, 0.1; plus 150, 0.1; minus 150, 0.2.

With water being added to the hydroseparator the ratio of water to perlite milled in tons for the entire operation is approximately 15 to 1.

It may be of interest to many operators to note that an average of 55 tons ground every 16 hr is maintained by the 2 by 6 rod mill, normally rated at 1 tph and that rod wear since the beginning of operations has averaged slightly less than 0.5 lb of rod per ton of rock milled. During the first several months of pilot work the wear was considerably higher than the average due to grinding tests at higher densities.

Processing

From the mine at Frieda, Oregon, the crushed perlite is shipped by rail to the processing plants, one at St. Helens, Oregon, another at Grand Rapids, Michigan. There the perlite sand is dried to facilitate handling whereupon it is injected into a horizontal, rotary kiln and subjected to intense heat, approximately equivalent to its melting point. Upon being suddenly brought to this temperature the sand expands to approximately seven times its original volume. From the kiln the popped perlite passes through a series of multicones for cooling and for collecting, dust laden gases passing through a water bath before being expelled and the perlite being drawn into 4 cu ft bags which are sewed ready for shipment or for further processing in the form of ready mixed, fibered, hard wall plaster or ready mixed accoustical plaster. Weight of the multiwall bags of expanded perlite is maintained at 40 to 44 lb per bag, 10 to 11 lb per cu ft.

Heat must be closely controlled during the drying to prevent "burning" of the perlite, removal of an excessive amount of combined water or possible disturbance of the physical character of the rock which renders it useless, resulting in discoloration and excessive weight in the end product. Laboratory tests indicated 600°F as the maximum safe temperature for drying, but actual operation showed that temperature to be too high. Trial and error methods reduced the maximum temperature to 300°F.

A delicately controlled feed is essential. Overloading of the kiln results either in perlite passing through without expanding or, more serious to the operation, in the perlite particles fusing to each other and to the lining of the kiln necessitating costly shutdowns. Underloading, on the other hand, results in loss of valuable production, or, again, in fusion and slagging because of improper dissipation of heat.

Temperature control is of the same importance as the feed control for much the same reasons. Too low a temperature results either in expansion failure or in loss of production. Too high a temperature results in fusion and slagging.

Integrally associated with the feed and temperature are the pressures within the kiln. Rate of flow, the product of the three, is indicated by the fact that within 1 min after the perlite sand has entered the furnace it is in the bag as expanded perlite, having been cooled sufficiently between the actual discharge point on the kiln and the bagger so as not to burn the bags. Perlite currently is processed at the rate of two tons per hour.

Cooling and the all important sizing largely are accomplished by air through a series of multicones between the furnace and the bagger from the first of which is drawn off the standard aggregate and from the second of which is drawn off a product referred to as "cone." Depending on the screen analysis of the standard, a portion of the cone which consists of fines may be added to the standard. The unused portion of cone which amounts to approximately 25 pct of the entire production remains a problem for sales and for research. Except for the small amounts that may be sold for trowel coat in the plastering industry the cone is wasted at present.

Perfect coordination must exist between the fireman and the bagger who, weighing every fifth bag and examining the expanded perlite as it is being bagged, immediately notifies the fireman of changes in weight or indications of burning or fusing in order that proper furnace control can be maintained.

Strength and size specifications for expanded perlite directly reflect the particular market for which it is intended. The plastering field, requiring

the least reprocessing, constitutes the largest market. Consequently in many sections of the United States the perlite aggregate conforms to the ASTM specifications for sizes of sand in plaster aggregate. In the concrete field local building codes dictate the strength requirements, but the demands of the contractors largely determine the particle size. Since the strength and weight are dependent primarily upon the raw material care must be taken in selection of the perlite deposit. It has been noted that perlites having a high water content and fine perlitic texture often expand more than those having a low water content and poor perlitic texture and that the particle strength is inversely proportional to the rate of expansion.

A typical daily average screen analysis of plaster aggregate from the St. Helens plant is shown in Table 5.

Table 5 . . . Screen Analysis of Plaster Aggregate

Cumulative Percent Retained on Screen
August 16, 1948

Mesh	Per Cent	ASTM Specification	
		Maximum	Minimum
+ 4	0.00	0	0
+ 8	0.09	10	0
+ 14	9.32	80	15
+ 28	52.60		
+ 48	79.85	95	70
+100	96.31		95
+200	96.06		
−200	100.00		

Average weight per bag, 41.0 lb.

As improvements were made in the efficiency of the pilot processing plant the amount of salable product became large enough to necessitate a regular sales department which was in operation, therefore, even before the plant had outgrown the pilot stage. Current production of the processing plant already has been stated as being 2 tph. The production figures for the period January 1 to July 1, 1948 are given in Table 6.

The principal market in the northwest, to date, is plaster aggregate. During the first half of 1948 expanded perlite replaced approximately 300 carloads of sand with plaster aggregate and concrete aggregate heading the list of sales. Second on the list was ready mixed accoustical plaster and third, ready mixed hardwall plaster. A noncombustible accoustical tile, still in the pilot plant stage of production, accounted for minor sales.

Outside of the plastering industry, expanded perlite is being used extensively for dry kiln insulation and for roofing insulation slabs. For the dry kilns the slabs are 6 in. thick with a 6 or 7 to 1 mix with portland cement. K factor for the loose aggregate is equivalent to that of ground cork, varying for the slabs according to the amount of binder or cement used. Other applications include soil conditioner, sandtone paint, scouring powder in soap, and filter aides.

Summary

With the widespread occurrence of perlite and similar volcanic glass deposits throughout the west and the southwest and with the existence of profitable local markets for the aggregates produced, there is much room for the expansion of processing plants now in operation as well as for the starting of new plants.

New operators can learn much from those already in production, an advantage that most of the present operators did not have when they started. Through underground development and research at the Lady Frances mine in Oregon, it has been found that intrusive, rhyolitic bands, commonly found in perlite deposits, may offer problems in beneficiation or selective mining, perhaps, even appreciably affect the estimated reserves, and that where underground mining is contemplated care must be taken in the choice of mining methods. Specifications for aggregates on the basis of current markets are such that high efficiency must be maintained throughout the milling and processing and care must be shown in the choice of the deposit to be worked.

The research and sales departments must cooperate to create additional markets, whether through new products for existing markets or through new markets for existing products, to reduce the actual loss resulting from unsalable fines.

Table 6 . . . Production Record for Period January 1 to July 1, 1948

Month	Ore Used (Tons)	Aggregate (Bags)	Cone (Bags)	Total (Bags)	Bags per Ton Raw Perlite
January	349	17,450		17,450	49.9
February	310	14,482	547	15,029	48.4
March	611	24,769	6,472	31,241	51.1
April	414	18,024	3,792	21,816	52.5
May	511	20,997	978	21,975	43.0
June	372	14,192		14,192	38.1
Totals	2,567	109,914	11,789	121,703	
Average					47.1

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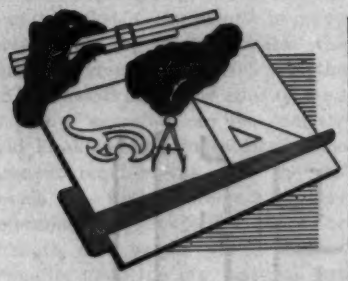

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MINING ENGINEER, preferably single, at least two or three years' experience on underground engineering work including mapping, to do engineering work, keep maps up to date, and assist in monthly reports from the property. $3600 to start, plus room, board, transportation. Central America. Y2291.

TEACHING PERSONNEL. (a) Instructor or Assistant Professor, preferably with Ph.B. and some industrial experience, to teach chemical engineering. Salary and rank open. (b) Instructor, recent graduate, at least M.S. degree, to teach chemical engineering. About $3800. (d) Instructor or Assistant Professor, preferably Ph.D. degree and industrial experience, to teach ferrous metallurgy. Rank and salary open.

ENGINEERS. (a) Mining Superintendent, 35-45, at least five years' supervisory experience including mine development work for feldspar, tin and gold project. $6000-$7200. (b) Chemical Assayer with some metallurgical accounting experience, to take charge of sampling and laboratory analyses of mine ores. $4200. South Dakota. Y2299C.

MINING ENGINEER, recent graduate, for gold mining operation. $2400 plus room and board. Colombia, S. A. Y2349(b).

ASSISTANT PROFESSOR to take charge of newly created curriculum in petroleum production engineering, southeastern university. $3900 for nine months. Possibility of $900 more for summer work in research. Reply by letter stating personal data, education, experience. Y2378.

JUNIOR MINING ENGINEER, graduate, with minimum of one or two years' practical experience in surveying and mapping. Should be qualified to establish and develop his surveys and maps from the beginning. Excellent opportunity with company engaged in large development program involving change of present system of mining and installation of entirely new equipment and methods. Starting salary, $3000-$3900. Pennsylvania. Y2541.

MINING ENGINEERS. (a) Mine Superintendent with working experience in Spanish speaking countries and familiar with variety of modern mining methods and possess a very good record in handling laborers effectively. Will be required to expand production and place operations on more efficient basis. $10,000. (b) Assistant Mine Superintendent with same requirements as above, but with less responsibility. $6000. (c) Chief Engineer for mine engineering department. $4800. (d) Mine Foremen, 2, one with block caving experience. $5000. Three-year contracts. Transportation and traveling expenses paid. Must have working knowledge of Spanish. Bolivia. Y2570.

MINING ENGINEER, 30-35, graduate, with general experience covering surveying, mapping, development, production, examination of prospects, etc., to keep maps and records up to date, check equipment specifications, geology reports, metallurgical accounts, etc., in general office. $5000-$6000. Headquarters, New York, N. Y. Y2589.

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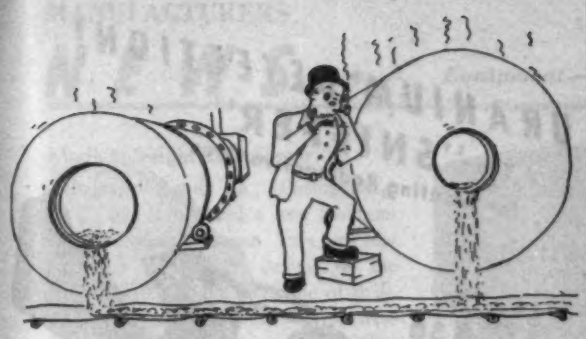
MINING ENGINEER, 33, married, technical graduate. Ten years exploration, shafts, tunnels, core drilling, and detailed reports. Supervised several major diamond drilling investigations. Presently employed as tunnel foreman on large project. Desires permanent re-



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CONSULTING ENGINEER, 55, record of proven accomplishments; thirty years' experience prospecting, mining, milling, ore buying metallic and non-metallic minerals. Married, now employed but available 30 days' notice. Desires full or part-time employment as executive or consultant. Location and travel not limited. M-448.

GEOLOGIST, B.S., M.S., recent graduate, single, 24, desires position with oil company, mining company, or geophysical establishment. Research or field work. Will go anywhere U. S. or Canada. M-449.

ORE DRESSING METALLURGIST, B.S., E.M., 30, married, one child. Eight years' varied experience, research, testing and production; flotation, gravity concentration, leaching; precious metals, rare metals, base metals. Supervisory positions held. Desires responsible position operations, research, teaching. Presently employed; available 2 to 4 weeks' notice. Prefers North or South America. M-450.

ECONOMIC GEOLOGIST, MINING ENGINEER (Registered P. E.) 39, married, Canadian and U. S. universities, mining geology and business administration. Broad experience mineral exploration, production geology, evaluation, supervisory experience major companies. Seeks responsible position exploration, development or consulting and mineral evaluation on part or full time basis. M-451.

MINING GEOLOGIST, 35, report single, Ph.D. Seven years' experience mining geology and engineering, U. S. and Latin America. Speaks good Spanish. Willing to travel. Available immediately. M-452-4510-E-1-San Francisco.

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Box F-19—MINING ENGINEERING

ATTENTION MANUFACTURERS!

Please see right-hand page 41 ⟶

Send us NEWS of interest to mining engineers for this well-read department.

XUM

Medium-weight Jackhamer

Ingersoll-Rand Co., Phillipsburg, N. J., has introduced a new medium-weight J-40 Jackhamer. This new machine is similar in design to the heavier J-50 model. Capable of drilling in any kind of rock, it is particularly well suited for general utility service in mines, quarries, and road work.

The "SNIFFER"

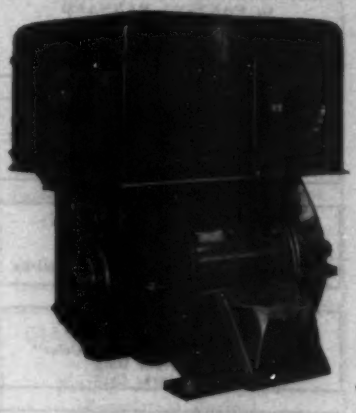
The NUCLEAR "Sniffer" is a small, sensitive Geiger counter for locating radioactive minerals. It will detect most radiation reaching the surface of the ground, regardless of how deep the source may be. It is light in weight—only two pounds, low in price, small in size, and most economical to operate. Ordinary flashlight batteries are the only power source needed. For further information, write to The Radiac Co., 489 Fifth Ave., New York 17.

Car Shaker

Allis-Chalmers Mfg. Co., Milwaukee, has announced a new car shaker for emptying coal, cinders, ore, slag, coke, sand and gravel and other granular materials quickly and safely from drop-bottom gondola cars. The new shaker incorporates such features as a stress-relieved body, and a completely new design mechanism with a minimum of working parts. It is expected to find a ready market wherever a saving of time and labor is desired as well as improved safety conditions in the continuous unloading of drop-bottom gondola cars.

VEE PLY Belting

Worthington Pump and Machinery Corp., recently added VEE PLY belting, the quick detachable V belt, to its V-Belt drive line. Specially designed to service emergency breakdowns, and eliminate a large, endless belt inventory, VEE PLY can be adapted to most V-Belt drive applications. Worthington VEE PLY belting is flexible and presents no problem of internal friction; thus maximum horsepower ratings are accomplished at minimum sheave diameters.



G-E Standard Unit-Cooled D-C Motors Available

General Electric has announced its standard totally-enclosed, unit-cooled d-c motors in ratings up to 200 hp. with a new type cooling assembly which enables them to operate at slow speeds. Two blowers, driven by a single G-E Tri-Clad induction motor, in the air-to-air cooling assembly, provide ventilation independent of motor speed. One blower circulates internal air through the motor and unit cooler, while the other blows external air through the cooler. Designed for operation in dusty, dirty, and oil-laden atmospheres, G-E totally-enclosed, unit-cooled motors require no piping, duct-work, air filter, or pressurized air supply.

Belt Conveyor Carrier

Stephens-Adamson Mfg. Co., Aurora, Ill., has announced a new SEALMASTER ball bearing belt conveyor carrier which combines many important design and construction features in a single unit, including strength, lightweight construction, low-power requirement, minimum maintenance, one-shot low-pressure lubrication, interchangeable parts, no tools or adjustments required. SEALMASTER ball bearing belt idlers with live shaft rollers set a new standard for economical belt conveyor operation. SEALMASTER carriers have been designed for simplicity, service and economy . . . they are lightweight, yet strong enough for continuous, heavy duty service.

NEWS

The Hardinge Co., Inc., York, Pa., has secured the manufacturing and sales rights to an accurate and rugged sampler—the "Hardinge Automatic Sampler." It is particularly adapted to the mining, stone products, ceramics, chemical, and allied process industries.

A new Edison electric cap lamp for miners has been developed by scientists at the General Electric Co., Thomas A. Edison, Inc., and the Mine Safety Appliances Co. It is claimed that this lamp gives 25 per cent more light than ever before available from a cap lamp, and it has been tested and approved by the U. S. Bureau of Mines.

The Dorr Co. has announced a new and distinctly different Dorr classifier, "type H," which incorporates a radically different head motion design and rake path and represents, it is said, a major step forward in mechanical refinements. This classifier is offered to the field as a fully-developed, commercially proved unit. Four years of mechanical development preceded rigid shop and field testing programs last year. Ten units are at present in full commercial use in American copper mills and 25 more are at present on order for other properties, both here and abroad.

National Mine Service Co. has announced the completion and full occupancy of its new, modern headquarters at Beckley, W. Va. This company, manufacturers of replacement parts for mining equipment, distributors of original parts for Joy, Sullivan, La-Del equipment, builders of mine locomotives and rebuilders of mining equipment, are the exclusive distributors of Wheat "Forty-Niner" electric cap lamps, National Model Koehler flame safety lamps, Hayden belt fastening equipment; they maintain electric and mechanical repair service, and carry large stocks of mine supplies.

(*Continued on p. 42*)

(*Continued from p. 41*)

V. L. Snow, who joined the Engineering Department of The Euclid Road Machinery Co. in 1935, has been promoted to domestic sales manager. Mr. Snow has devoted much of his time in recent years to the development and application of Euclid equipment on the Minnesota Iron Range. Succeeding Mr. Snow as assistant sales manager is *J. W. Bloomquist* who joined Euclid as district representative for the Minnesota territory in 1945.

Bulletins

Rotary Pumps. Worthington Pump and Machinery Corp., Harrison, N. J., has a new line of rotary pumps known as GAMI. The new pumps employ the use of a mechanical seal and a self-lubricating type inboard sleeve bearing to carry overhung drives. Write for bulletin W-484-B-1.

Float-Sink Concentrator. The Link-Belt will be glad to send, on request, a copy of Catalog No. 2101, describing the float-sink concentrator which has been successfully used in coal cleaning and can be adapted to the concentration of other non-metallic or metallic ores that are amenable to separation by gravity methods.

Air Hose. A new catalog section on its line of air hose has been published by The B. F. Goodrich Co., Akron, Ohio. The catalog section features the company's "Highflex" brand, an improved type introduced recently which the manufacturer says is 50 per cent lighter than conventional air hose of the same working pressure.

Industrial Power. "Industry Uses Caterpillar Diesels" is the title of a new 12-page booklet, Form 12113, issued by the Caterpillar Tractor Co. Graphically and pictorially illustrated, this publication presents an informal question-answer format on the various problems confronting industrial power usage in the equipment field.

Layne Pipe Orifice Meter Method of Computing Water Flow. Every engineer who has occasion to measure the flow of water from an open pipe should read "Measurement of Water Flow Through Pipe Orifice with Free Discharge." This handy pocket sized booklet has been revised and re-issued by Layne & Bowler, Inc., Memphis, Tenn.

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(*Continued from p. 18, Section 1*)
lector's item. In addition, a number of outstanding experts (named on p. 64) gave their time willingly to assist in clearing up some of the knottier problems on the "Bergbüchlein."

The book is also of interest because it shows how little human nature, especially in the mining field, has changed in 400 years. Witness the following, spoken by young Knappius, the apprentice miner, who was being instructed by Daniel, the expert: "I want to ask this: In what direction or face in the shaft or at what corner of the drift would my . . . share be located?—so that I can see in the pit what profit I can expect." No wonder Daniel replied "Your ignorance about mining has driven me to undertake this job . . . If you value gain more than the art, you will have to do without either art or profit."

Both Mrs. Sisco and Dr. Smith have done a splendid job in putting these little books into clear but flavorsome English, and the Mudd Fund is to be congratulated for underwriting their publication. The format was designed by Carl Purington Rollins and one thousand copies were printed by the Yale University Press; so only one AIME member out of twenty can own this book ($3.00 to AIME members, $5.00 to others.) An early order is thus clearly indicated.

—*Frank T. Sisco.*

•

Safety Lamp. The new, lightweight "National" Model Koehler flame safety lamp represents an advance in flame safety lamp design. This model weighs only 1 lb. 12¼ oz. and measures 8¾ in. from the base to the top of the dome. It is light to carry and easy to use when working in low coal. The lamp is a very compact unit, strongly constructed of durable aluminum alloys to withstand severe operating conditions. It is not affected by moisture or dampness; will not rust or corrode. This safety lamp is distributed exclusively by National Mine Service Co., with divisions in Beckley and Logan, W. Va.; Forty Fort, Pa.; Jenkins and Madisonville, Ky.

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MINNEAPOLIS, MINNESOTA, U. S. A.
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